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FOREWORD

The 190th Volume of the Transactions of the American Institute of Mining and Metallurgical Engineers, consisting of technical papers and discussions published in 1951 in the large field encompassed by the Mining Branch is presented herewith. The individual fields that are covered in this volume are Metal Mining (twelve papers), Mineral Beneficiation (twenty papers), Coal (nineteen papers), Industrial Minerals (thirteen papers), Geology (six papers), Mineral Economics (one paper), and Geophysics (three papers), plus three technical notes and four discussions of papers published prior to 1951.

In consulting the index or the table of contents to this volume, please note that the interruption in pagination is caused by the fact that all of the articles have appeared previously in Mining Engineering which was published in twelve separate issues throughout 1951 with consecutive pagination.

In view of the fact that many members of the Mining Branch are interested in those papers dealing with Extractive Metallurgy, the nineteen papers dealing with that subject, which are published in the Metals Transactions volume, No. 191, are listed for reference purposes at the end of the table of contents of this volume.

The excellence of the volume is due in large part to the critical screening and selection of material by the various program and auxiliary publications committees. Thanks are due these committees for their diligent work. In addition the Mining Branch expresses its appreciation to the various authors for their efforts in preparing the papers for presentation and later for publication by the Institute. The wide range of material that is covered reflects the great scope of interest of the Institute as a whole and specifically of the Mining Branch.

The second year of the Mining Branch as an integral unit of the Institute is now drawing to a close. The strength of the Branch is due chiefly to the widespread and varied activity of the four major Divisions. During the past year these Divisions have sponsored a large number of interesting meetings in many parts of the country, thereby creating the possibility for large numbers of technical people to share their knowledge, to form new and profitable associations, and to participate actively in the affairs of the Institute. Through such meetings and associations a large part of the potential value of the Institute is being realized. For the success of these many activities, special recognition should be made of the vigorous leadership of each Division by its chairman and the chairmen of the respective program committees.

Richard M. Foose, *Chairman*
Mining Branch Council

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The Probability Theory of Wet Ball Milling and Its Application. By E. J. Roberts. (1950) **187**, p. 1267 988
Progress Report on Grinding at Tennessee Copper Co. By J. F. Myers and F. M. Lewis. (1950) **187**, p. 1133 990
The Cleaning of Fine Sizes of Bituminous Coals by Concentrating Tables. By R. E. Zimmerman. (1950) **187**, p. 956 1000

Extractive Metallurgy

As many members of the Mining Branch are interested in the papers of the Extractive Metallurgy Division, the titles of the papers published for that Division in 1951 are listed below. These papers, and discussion, appear in Transactions (1951) **191**, Metals Branch.

Industrial Hygiene at American Smelting and Refining Company. By JOHN N. ABERSOLD and K. W. NELSON.
Sintering Practice at Josephtown Smelter. By H. K. NAJARIAN, KARL F. PETERSON, and ROBERT E. LUND. With discussion.
Preparation of Titanium Tetrachloride from Rutile. By C. H. GORSKI.
Reverse Leaching of Zinc Calcine. By L. P. DAVIDSON, R. K. CARPENTER, and H. J. TSCHIRNER.
Metallurgical Reactions of Fluorides. By H. H. KELLOGG. With discussion.
El Paso Refinery of Phelps Dodge Refining Corporation. By B. B. KUNKLE.
Sulphur Activities in Liquid Copper Sulphides. By R. SCHUHMANN and O. W. MOLES.
Vacuum Treatment of Parkes' Process Crusts on a Pilot-Plant Scale. By A. W. SCHLECHTEN and R. F. DOELLING.
Extraction of Alumina from Haiti and Jamaica Bauxites. By T. D. TIEMANN.
Thermodynamics of Iron-Silicate Slags: Slags Saturated with Gamma Iron. By R. SCHUHMANN, JR. and P. J. ENSIO. With discussion.
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Low Grade Bauxites and Clays as Potential Aluminum Resources. By K. K. KERSHNER, C. W. FUNK, and W. A. CALHOUN.
Rate of Evaporation of Zinc at Low Pressures. By H. W. ST. CLAIR and M. J. SPENDLOVE.

Classification of Block Caving And Draw Methods

by F. S. McNicholas

Various methods of block caving and draw practice are classified and the advantages, disadvantages, rock, treatment, size of orebody, costs, and profit comparisons, and other factors which determine the selection of the proper method are outlined.

IN the writer's opinion the term "block caving" has been rather loosely applied to a general principle of mining, and therefore detailed classification of caving methods and draw, together with the characteristics required for each method, will fill a desired need.

Selection of caving method should be made only after a study of all governing conditions, such as size of orebody, lateral and vertical extent of orebody, attitude, treatment demands, possible resultant pressures, and cost and profit comparison of methods.

Successful caving methods can be classified under six main headings: (1) Full-protected block caving, (2) semi-protected block caving, (3) unprotected block caving, (4) full-panel caving, (5) half block-panel caving, (6) block-panel caving. The methods and factors governing choice of method are shown in Table I.

Full-protected Block Caving

Full-protected block caving consists of developing, undercutting, and drawing to exhaustion a block of ore of predetermined size without coordination of mining or draw with adjoining areas. The block is mined and drawn as an independent unit (Fig. 1).

Full-protected block caving demands that mining and draw cannot be done against unconsolidated capping in adjoining caved and exhausted blocks. Two sides of block are against solid. Caved capping must be allowed to consolidate for a suitable period, usually not less than one year. Pillars may be left in lieu of consolidated capping (Fig. 2).

The advantages of full-protected block caving are: (1) A definite area is developed, mined, and drawn as a unit so that all work may be completed, the block drawn to exhaustion in the minimum length of time, and the area then abandoned. (2) Production is not restrained because of coordination with adjoining areas. (3) Records of draw and caved reserve of ore are good. Upon abandonment of the block, accountability for production is closed out with no carry-over to adjoining areas.

The disadvantages of full-protected block caving are: (1) Side dilution from the draw adjacent to the consolidated capping may be serious, particularly where capping does not consolidate sufficiently in the allotted time. (2) Boundary weakening is needed on the solid sides of the block except in weak rocks. (3) Maximum cantilever and compression effect to aid caving and fragmentation may not be secured since the draw should proceed from the solid to the consolidated capping to minimize side dilution. However, undercutting from the loose (caved capping) to the solid with a slight initial draw in the same manner will develop cantilever effect. (4) A period of at least one year, more is usually desirable, must be allowed for caved capping to consolidate before mining the adjoining block, depending upon the characteristics of the capping. (5) Capping may not consolidate sufficiently in the time that can be allowed.

Full-protected block caving is indicated where: (1) The ground is weak and heavy and workings

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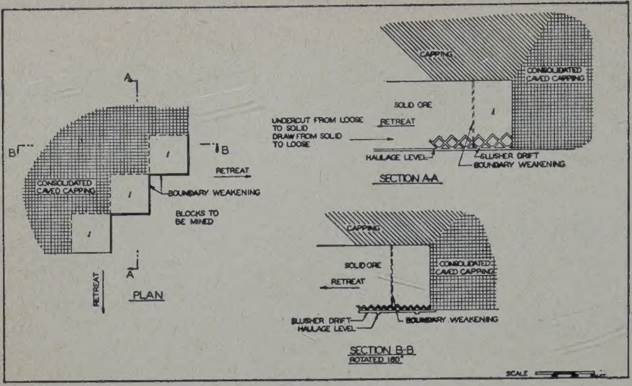
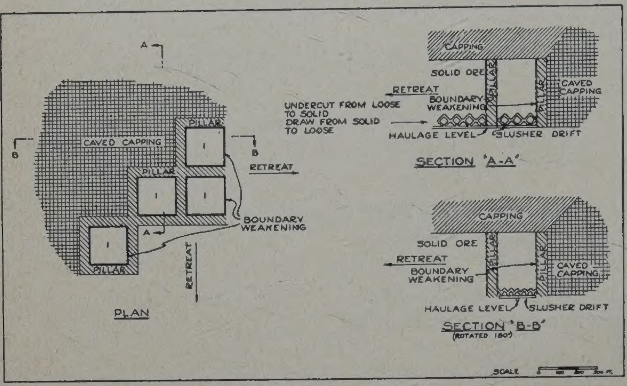
Discussion on this paper, TP 2987 A, may be sent (2 copies) to AIME by Feb. 28, 1951. Manuscript, Jan. 27, 1950. New York Meeting, February 1950.





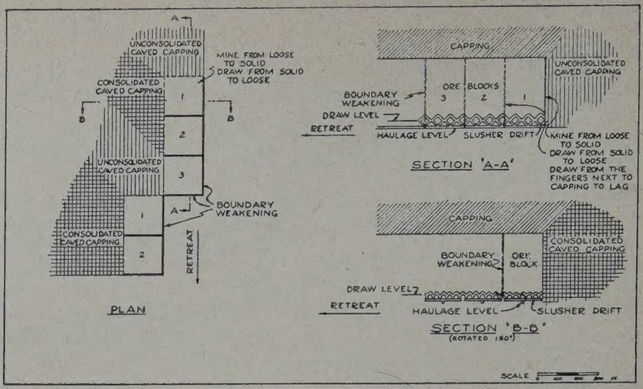


Fig. 1 (upper left)—Full-protected block caving. Fig. 2 (lower left)—Full-protected block caving with pillars. Fig. 3 (right)—Semi-protected block caving.

are difficult and expensive to establish and maintain, and therefore maximum speed of working out a block is necessary. (2) The broken ore consolidates in a short period. (3) The caved capping consolidates firmly in a short period. (4) Different types or grade of ore are desired. (5) Oxidation of caved ore is a serious treatment problem. (6) Ample mining area is available. (7) The capping is waste.

Semi-protected Block Caving

Semi-protected block caving is similar to full-protected block caving except that the block has one side against consolidated capping, one side against newly caved capping, and two sides against the solid, thus allowing the block pattern to retreat along a panel and avoiding to some extent the offset checkerboard pattern of full-protected block caving in one direction. This method is shown in Fig. 3.

The advantages of semi-protected block caving are the same as full-protected block caving plus: (1) A more regular retreat pattern in line is possible. (Retreat is along a panel). (2) The time schedule is more regular as time for consolidation on one side only must be allowed.

The disadvantage of semi-protected block caving is that more side dilution is likely from the side against unconsolidated capping.

Semi-protected block caving is indicated in the following instances: (1) The ground is weak and heavy and workings are difficult and expensive to establish and maintain, and therefore maximum speed of working out a block is necessary. (2) Broken ore consolidates in a short period. (3) Caved capping consolidates firmly in a short period. (4) Oxidation of caved ore may result in a serious treatment problem. (5) Ample mining area is available. (6) The capping is low-grade ore, and the cut-off grade is relatively high.

Unprotected Block Caving

Unprotected block caving is similar to full-protected block caving except that the block has two sides against the solid and two sides against unconsolidated capping (Fig. 4).

The advantages of unprotected block caving are: (1) Extreme flexibility. (2) No time is needed to allow the capping to consolidate. (3) Almost any sequence of mining blocks may be followed that will maintain two sides against the solid.

The disadvantage of unprotected block caving is that side dilution from two sides against unconsolidated capping may be serious.

Unprotected block caving may be indicated where: (1) Maximum speed of developing, mining and drawing an area is desired. (2) Low cost of developing, mining and draw is necessary. (3) Capping is low-grade ore, and the cut-off grade is relatively high (dilution is by low grade of near break-even grade). (4) Low-grade ore mass is to be broken up for a cheap secondary method of recovery such as leaching in place. (5) Cantilever effect is not necessary to cave and fragment the ore. (6) Capping breaks much coarser than the ore, and dilution is thus minimized. (7) Mining area is limited. (8) Broken ore consolidates in a short period.

Full-Panel Caving

Full-panel caving consists of developing and undercutting adjoining blocks of ore in a regular retreating manner with the draw following the mining sequence and regulated to give a retreating draw plane which maintains a toe of broken ore against the block being mined on those sides which have been mined. No block boundary cut-offs or weakening is used. Cut-offs are needed on the periphery of the orebody and to initiate caving.

Almost any sequence of mining adjoining blocks may be followed as dictated by the plan of retreat but should be followed wherever possible in a manner to avoid having block with more than two sides against caved material (Fig. 5).

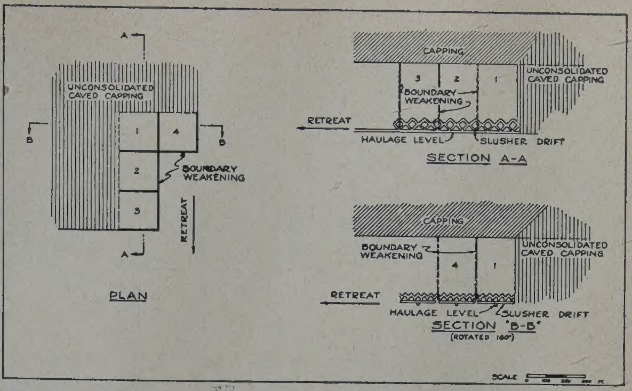
The advantages of full-panel caving are: (1) The benefit of cantilever and compression action to aid caving and fragmentation may be secured. (2) Boundary cut-offs and weakening are reduced since cut-offs or boundary weakening are needed only along footwall of the orebody (and perhaps to some degree in hanging wall) and to initiate caving action. (3) It is a very flexible method as a predetermined sequence of mining of blocks usually may be altered somewhat without bad effect. However, this change in sequence may be *local only* as the general established sequence must be followed. (4) There is no time lag awaiting consolidation of caved capping. (5) A very regular sequence of retreat may be established and followed. (6) Minimum cost is secured.

The disadvantages of full-panel caving are: (1) The proper sequence and regulation of the draw is difficult to achieve and maintain and may tie up large potential draw areas reducing production dangerously. Loss of draw points and workings tied up for repair, etc., influence large areas. (2) Since the draw is necessarily from the loose to the solid, good control is essential to prevent funneling through the protecting toe of broken ore into the unconsolidated capping. (3) Large draw areas are required for a given tonnage due to draw coordination with adjoining tonnage and to secure cantilever effect. (4) There is considerable tonnage of broken ore that is unavailable in the protecting toe. Development and equipment for this tonnage is also tied up. (5) Since accountability for remaining broken ore reserve is by the depletion method and cannot be confined to a given area, all of the possible serious errors are thrown into remaining broken ore reserve. These errors may be cumulative, and over a period of time the record of remaining broken ore reserve may be in serious error.

This method should not be used in heavy ground where maximum speed of working out a block is necessary, or where cantilever effect would throw undesirable weight on adjoining areas to be developed and mined. It is applicable where: (1) The ground is medium strong to strong, where maintenance of workings over a considerable period is not a serious consideration, and where maximum cantilever effect is desired, and speed of working out a block is not important. (2) Neither caved capping nor caved ore consolidate to an important degree. (3) Oxidation of caved ore does not present a serious treatment problem. (4) Only one class of ore is to be mined and the grade of ore drawn is limited to the possible variation of draw under the draw control plan. (5) Large mining area is available. (6) Minimum cost is desired.

Half Block-Panel Caving Method

This method, shown in Fig. 6, consists of developing, undercutting, and drawing blocks of ore with two sides against the solid, one side against consolidated caved capping, and the remaining side cushioned from unconsolidated caved capping by a "toe" of broken ore. The retreat may be continuous in one direction and is away from unconsolidated caved capping.



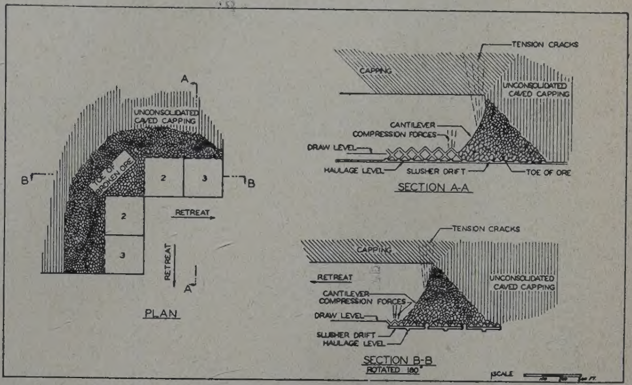


Fig. 4 (top)—Unprotected block caving.

Fig. 5 (bottom)—Full-panel caving.

The advantages of half block-panel caving are: (1) The retreat is mainly in one line and is with minimum coordination of draw with any adjoining blocks, so speed of mining the panel could be the maximum if retreat is along the strike. (2) A more regular pattern of retreat is possible. (3) Maximum benefit of cantilever and compression action to promote caving and fragmentation may be secured. Said forces are confined, to some extent, to one line and may be observed and controlled to secure maximum effect. (4) Boundary weakening is needed on one side only in addition to the undercut. (5) Combined top and side dilution should be minimum. (6) There is coordination of draw of fingers in the panel but not with any adjoining panel. (7) There is a minimum of broken ore tied up in the protecting "toe" on one side only. (8) Accountability superior to any other panel draw.

The disadvantages of half block-panel caving are: (1) If the panel retreat is down dip of a steeply dipping orebody, and the draw is from the loose to the solid, and if there are irregularities in the strength of the orebody, there may be a tendency to form cantilevers up dip and allow capping to draw under ore. If the panel retreat is along strike this does not hold. (2) The method demands consolidated capping on one side to avoid excessive dilution which would result from unconsolidated capping. (3) Mining area may be limited since the succeeding panel must await consolidation of capping.

Half block-panel caving is indicated where: (1) The ground is good and maintenance over a medium term is not a serious problem. It may be used where maximum speed of mining an area is desired. (2) Ore is of good grade and capping is waste. (3) *Controlled* cantilever and compression effect is desired with minimum arching effect. (4) Natural character of orebody favors retreat and cantilever effect is one line, or direction. (5) Retreat is along the strike of a steeply dipping orebody. (6) Caved capping consolidates in a reasonable period of time.

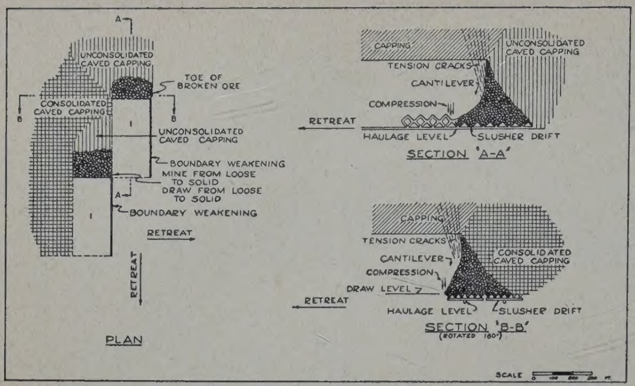
Block-Panel Caving Method

This method consists of developing and mining adjacent blocks of ore as units in a retreating manner with the main draw of the block from solid to loose and regulated to maintain a relationship with adjoining blocks which are being drawn. Cut-offs weaken the boundary on the two sides next to the solid. Undercutting proceeds from the loose to the solid and a slight draw follows the undercutting to establish cantilever effect. After undercutting is completed, the main draw is from solid to loose

Table I. Block Caving Methods and Factors Governing Choice of Method

Method	Rock Characteristics			Working Installation and Maintenance	Speed of Exhaustion of Block	Mining Area and Method	Treatment Demands	Dilution	Recovery	Draw	Accountability	Costs
	Ore 1	Capping 2	Footwall 3	4	5	6	7	8	9	10	11	12
Full-protected Block Caving	Weak Heavy Breaks fine Contains binder, damp. Consolidates in short period. Cantilever effect not necessary for caving and fragmentation.	Is waste Contains binder. Consolidates in short period. Caved capping adjoining blocks to be worked allowed to consolidate.	Weak Heavy Difficult and expensive to install and maintain workings.	Difficult and expensive to install and maintain due to 1 and 3.	Rapid exhaustion of block essential due to 1, 3, 4 and 7.	Method is indicated where mining area is not large. Undercutting from loose to solid. Two sides have boundary weakening.	Treatment demands different types of ore or a combination of grade. Oxidation of caved ore kept to minimum.	Dilution kept to reasonable amount.	Good to excellent with proper draw control.	No co-ordination of draw with adjoining blocks making rapid draw possible. Draw is from solid to loose.	Accountability of metal content the best. No cumulative errors, when block is completed.	Reasonably low costs but higher than full panel. Is a low cost mining method.
Semi-Protected Block Caving	Ore is weak Heavy Breaks fine Contains binder, damp. Consolidates in short period. Cantilever effect not necessary for caving and fragmentation.	Capping is low grade ore so dilution is with low grade not waste. Contains binder. Consolidates in a short period.	Weak Heavy Difficult and expensive to install and maintain workings and equipment.	Difficult and expensive to install and maintain workings and equipment due to 1 and 3.	High speed of exhaustion of block essential due to 1, 3, 4 and 7.	Method indicated where mining area is not large. Undercutting from loose to solid. Two sides have boundary weakening.	Treatment demands different types of ore or a combination of grade oxidation of caved ore kept to minimum.	Dilution from side against unconsolidated capping is greater than with a full-protected block. Dilution may be held to a reasonable amount by strict and proper draw control.	Fair to good with proper draw control.	No co-ordination of draw with adjoining blocks making rapid draw possible. Draw is from solid to loose.	Accountability of metal content is good. No cumulative errors when block is completed.	Is a low cost mining method. Reasonably low costs about same as full-protected block method.
Unprotected Block Caving	Ore is weak Heavy Breaks fine Contains binder, damp. Consolidates in short period. Cantilever effect not necessary for caving and fragmentation.	Capping is low grade ore of near breakeven grade. Is strong and breaks big and blocky so dilution may be low. Caved capping of adjoining blocks not given time to consolidate.	Weak Heavy	Speed of getting into and out of a block may make installation and maintenance of workings less costly than for full or semi-protected blocks.	Maximum speed of developing and exhaustion of block and area is secured.	Mining area may be relatively small. Undercutting from loose. Boundary weakening on two sides.	Low-grade mass is broken for a cheap secondary method of recovery such as leaching in place. Treatment demands different types of ore or combination of grades. Oxidation of broken ore a minimum.	Dilution may be high or if low recovery is acceptable, dilution may be reasonable.	Recovery will be relatively poor or high dilution must be accepted.	No co-ordination of draw with adjoining blocks. Rapid draw is possible. Draw is from solid to loose.	Accountability will be poor, but without cumulative errors.	Is very low cost method applicable to low-grade ore.

Full-Panel Caving	Ore is medium strong to strong. Maximum cantilever effect desired to cave and fragment ore. Caved ore contains little binder and does *not* consolidate to an important degree.	Caved capping contains little binder and does not consolidate to an important degree.	Strong to medium strong.	Cost of maintenance over a long period is not a problem due to 1 and 3.	Speed of exhaustion of block not important due to 1, 3 and 10.	Large area required due to 10. Undercutting from loose to solid. No boundary weakening. Cut off on perimeter of ore body only.	Only one class of ore can be produced. Grade of ore drawn is limited to possible variations of draw as dictated by the draw control plan. Method not suitable if oxidation of broken ore is a problem.	Dilution is a reasonable amount if proper draw controls are maintained. May be *less* than by other methods.	Recovery is good to excellent with proper draw control.	The full panel draw demands coordination with adjoining draw points. Draw is from loose to solid in retreating manner. Maintaining a retreating toe of ore under caved capping.	Accountability of metal content is very poor. The depletion method demanded to establish remaining broken ore reserve throws all errors into remaining broken reserves which over a period may result in drastic error.	Under proper conditions minimum costs will be secured.
Half Block-Panel Caving	Ore is medium strong to strong. Caved ore contains little binder and does not consolidate to an important degree. Maximum *controlled* cantilever effect can be secured.	Caved capping breaks big or contains binder and will consolidate.	Strong to medium strong.	Maintenance of workings is for a shorter period than for full panel caving.	Speed of exhaustion of block may be relatively rapid.	Mining area may be less than full panel and more than full protected block. Undercutting is from loose to solid. Boundary weakening on one side only.	Only one class of ore can be produced under ordinary circumstances. Limitations of possible grade are less rigid than full panel.	Dilution is a reasonable amount with proper draw controls.	Recovery will be good with proper draw controls.	The draw is half panel. Coordination of draw confined to fingers in the panel. Draw is from loose to solid retreating in one direction only. Protective toe of ore on one side of block.	Accountability of metal content is reasonably good. Accountability comes to a check upon completion of panel.	Low costs will be secured. Less than full protected block. More than full panel.
Block-Panel Caving Is a combination of unprotected block caving and a full panel cave and draw. Is designed to limit stresses to the block and limit the area necessary for full panel draw.	Ore may be medium strong to strong or weak. May or may not contain binder. May or may not consolidate in short period. Cantilever effect can be secured to some extent.	Capping may have almost any characteristics, preferable breaks big with little fines.	Footwall may have any characteristics that apply to caving methods. Preferably strong.	Maintenance of working and installation is for a relatively short period.	Speed of exhaustion of block is rapid. Speed of production from an area is maximum.	Maximum production secured from given area. Weight and stresses confined to the block. Undercut from loose to solid.	Only one class of ore can be produced. Oxidation of broken ore is kept to a minimum.	Dilution is reasonable to high. Proper rigid draw controls a must. After undercutting is complete, draw is from solid to loose with fingers against loose lagging 25 to 30 pct.	Recovery good to medium. Proper rigid draw controls a must.	Slight draw following the undercutting from loose to solid allows cantilever effect to develop. Main draw after undercutting completed is from solid to loose. This is *essential.*	Accountability of metal content is poor but better than full panel.	Costs will be more than full panel, about the same as unprotected block.



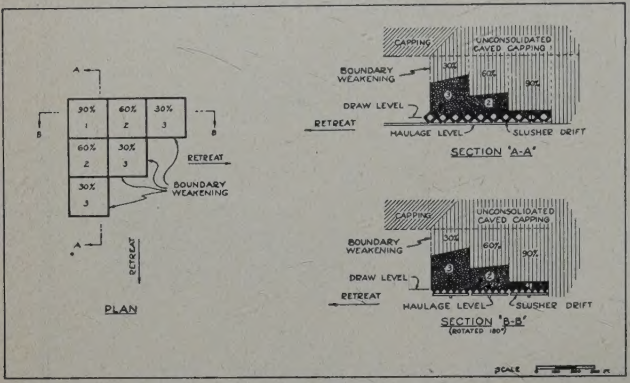


Fig. 6 (top)—Half block-panel caving.

Fig. 7 (bottom)—Block-panel caving.

with the fingers next to the loose lagging about 25 pct behind those next to the solid. When about 25 to 40 pct of tonnage is drawn, the entire ore in the block has caved and is about the same "fluidity" as the unconsolidated capping so further draw should be vertical.

The sequence of mining of blocks would be on an offset checkerboard pattern to give maximum production. This method is shown in Fig. 7.

The advantages of block-panel caving are: (1) Allow draw to retreat from the solid toward the broken ore cushion. With proper control of the draw in this manner, by the time the draw has progressed to the broken ore cushion, the ore in the entire block has caved and the density (fluidity) of the entire mass is about the same, so tendency of the draw would be to pull vertically. (2) Dilution and loss of ore should be less than most other methods excepting half block-panel. (3) Allows draw without time lag needed for consolidation of adjoining caved capping. (4) High continuous rate of production.

The disadvantages of block-panel caving are: (1) Full cantilever effect is not secured. (2) It requires weakening on two sides. (3) Draw is essentially a panel draw and if followed strictly will result in tying up large areas.

This method is indicated where: (1) The ground is strong to medium strong and maintenance of workings over a medium term is not a serious problem. (2) Maximum speed of exhausting an area is not essential. (3) Capping is strong and breaks bigger than ore with little fines. (4) Capping does not consolidate to a marked degree in a reasonable time. (5) Mining area is limited and maximum production must be secured from a given area. (6) Broken ore does not consolidate in a short period. (7) Draw must be from solid to loose to minimize dilution from adjoining unconsolidated caved capping or sand fill or other filled stopes.

Determination of Characteristics

Rock Characteristics: The characteristics of the rock of the orebody, capping and footwall may be determined by: (1) Detailed examination and classification of diamond drill core in respect to fracture pattern, fracture cleavage, jointing, texture, oxidation, mineralization, binder, etc. (2) Comparison of diamond drill core with core from orebodies of known characteristics by direct comparison of colored transparencies. (3) Detailed examination and classification of characteristics of excavation, and perhaps some trial blocks, compared with characteristics of excavation in known orebodies having known characteristics. (4) Comparison of colored transparencies of excavations comparing known characteristics with those under study. (5) Detailed study of caved ore and caved capping to determine size of fragments, amount of fines, amount of binder, amount of moisture, amount of consolidation over a period of time. (6) Application of observed general characteristics of known orebodies compared to the orebody under study.

Treatment Demands: May be determined by test work concerning the metallurgy involved, the desirable ratios of different ores or grades and the marketing demands.

Oxidation of caved ore may be a determining factor in arriving at the method to be used.

Mining Area and Speed of Production: Tonnage demands and speed of production from a block or an area may indicate the desirable method.

Draw: Experimental work on draw in scale models approaching actual proven conditions as closely as possible may give useful information, however, such work should have some background of practical operation to be of value.

Experimental work of others should be reviewed for possible application.

Experimental work will generally indicate a lower dilution than will be secured in practice.

Dilution and Recovery: It is difficult and perhaps impossible to predetermine these factors to an absolute degree. They may, however, be determined to a relative degree, and in such a manner be useful in determining method.

Accountability: Accountability of metal content probably will be of minor importance in determining method, however, small concessions may be made to secure good accountability.

Costs: All methods give reasonably low costs. Determined characteristics will largely determine method with its accompanying costs.

Acknowledgment

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Radiotracer Studies on the Interaction of Dithiophosphate with Galena

The Depressant Action of Phosphate

by G. L. Simard, M. Burke, and D. J. Salley

The depression of dithiophosphate by phosphate was investigated by correlating radiotagging measurements of competition in sorption with measurements of flotation. Competition in sorption was as complex as the sorption of the individual ions, thus strengthening the conclusion that more than one process was involved in the interaction of the ions with galena. The depression of flotation was chiefly a result of reducing the amount of dithiophosphate sorbed.

A PREVIOUS paper[1] described experiments on the interaction of dithiophosphate with galena and suggested that the process had characteristics of both chemisorption and surface chemical reaction. In order to add to this knowledge, a study has been made of the competition of a second ion with the dithiophosphate for a galena surface. Use again was made of radiotracer methods. Since radiophosphate was readily available, it was chosen as the competing ion. It is a feature of the tracer method that, in spite of similarity in the chemical behavior of the ions, the measurement of one ion in the presence of another can be made as simply as if only one ion species were present.

In order to study the competition effectively, it naturally was necessary to examine the interaction of phosphate with galena alone as well as in the presence of dithiophosphate; consequently, this report consists of two parts. The first is devoted to a study of the rates, the isotherms, the desorption, and the exchange in the phosphate-galena system in the same manner as was done for the dithiophosphate-galena system. The second portion is concerned with the studies on the competition. Flotation experiments on the three-component system are also outlined and the relationships involved are discussed briefly.

The experimental method was that of the previous paper. The preparation of the galena, the radioactive and the nonradioactive dithiophosphate, and the procedures for radioactive counting, for sorption, and for flotation measurements were identical to those previously employed. The radiophosphate* was used in the form of disodium phosphate solution.

In the competition experiments the effect of the phosphate ion in depressing dithiophosphate uptake was followed by holding the initial concentration of nonradioactive phosphate constant while varying the initial concentration of the radioactive dithiophosphate. Counterfashion, the depression of phosphate by dithiophosphate was obtained by keeping the initial nonradioactive dithiophosphate ion concentration constant and altering the initial concentration of the radioactive phosphate ion.

All solutions were buffered at a pH of 7.8 ± 0.1 by 20×10^{-8} mol per cc of sodium carbonate. Measurements with radiocarbonate indicated that little or no carbonate could be sorbed in the presence of

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* For early experiments (1943-1944), the elementary red phosphorus and the phosphate were obtained from the cyclotron group at the Crocker Radiation Laboratory of the University of California, Berkeley, through the courtesy of Dr. Joseph W. Hamilton. For more recent work (1946-1949), both radiophosphate and phosphorous have been supplied by the Oak Ridge National Laboratories on allocation from the U. S. Atomic Energy Commission.

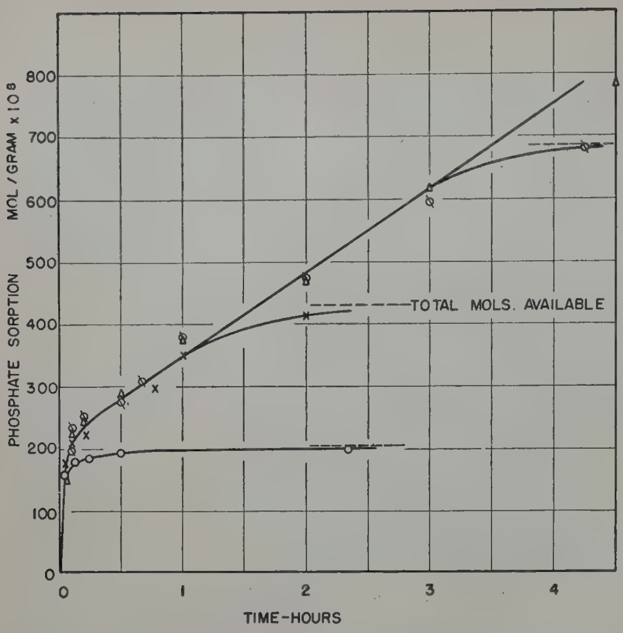

Fig. 1—Rate of phosphate sorption on unleached galena No. 3.
Initial phosphate concentrations: ⊙, $8.1 x 10^{-8}$; X, $17x0^{-8}$; ∅, $27x10^{-8}$; △, $50x10^{-8}$ mol per cc.

phosphate, although in its absence some sorption was possible.

Interaction of Phosphate and Galena

Rates: Figs. 1 and 2 show the essential facts concerning the rates of uptake of phosphate by galena. A rapid initial sorption occurred which was essentially complete in 10 to 12 min. In nitrogen no following secondary process took place on leached galena. However, in air there did occur a slower, secondary uptake which proceeded at a constant and identical rate on both unleached and leached galena. The rate of this secondary interaction was independent of concentration until the phosphate in solution was relatively low. After very long periods of time, the phosphate concentration asymptotically approached very low values, as is indicated in Table I. (The sorption in nitrogen was the same whether or not leaching had been done in the presence or absence of air).

Isotherms: The dependence on concentration of the amount of phosphate sorbed in the rapid initial process at the end of 12 min is shown by the typical results of Fig. 3 for ground mineral and for single crystals, respectively. In both cases sorption took place at the lowest concentration studied, $0.2x10^{-8}$ mol per cc.

Independence of concentration was generally attained at a low concentration of phosphate, and the low concentration region was not defined sufficiently well to permit fitting to a Freundlich type equation; this was in contrast to the case of the dithiophosphate isotherms. However, in similarity with the dithiophosphate observations, a considerable part of the sorption on unleached galena was by leachable material. Also, the scatter of the data for single crystals was again correlated with the appearance of stain on the crystal face.

There was a marked increase in sorptivity on exposure of the mineral to air. For example, on galena No. 5, $55x10^{-8}$ mol per g more phosphate was sorbed by leached mineral when it was dried in air subsequent to leaching than when the mineral was kept wet and sorption measurement made immediately. The amounts of phosphate corresponding to the horizontal portions of the 12-min isotherms for various unleached galena preparations are given in Table II and illustrate the variability between similar preparations of unleached mineral.

Desorption: Desorption measurements for galena No. 2 in water and galena No. 3 in $20x10^{-8}$ mol per cc Na_2CO_3 are given in Table III. The sorption times were 12 min for No. 2, and 26, 36, and 182 min for No. 3. The data for preparation No. 3 are average values from five measurements.

The desorption was unaffected by either the quantity of phosphate initially on the galena or the time used for its sorption. The desorption of phosphate from ground, unleached galena took place at a slower rate and to a lesser extent than the desorption of dithiophosphate.

Table I. Phosphate Concentration After Long Periods of Time Galena No. 3
($20x10^{-8}$ mol per cc Na_2CO_3)

Initial Concentration Mol per cc x 10⁸	Adsorption Time, Hr	Final Concentration Mol per cc x 10⁸
8.1	4	0.14
19.0	17	0.07
25.0	23	0.03
27.4	42	0.02

Table II. Phosphate Sorption by Unleached Galena at 12 Min

Galena	Phosphate Sorbed Mol per g x 10⁸
No. 2	240
No. 3	180
No. 4	850
No. 5 (unleached)	240
No. 5 (leached)	85
Single crystals	(50)[a]

[a] For 1000 cm² per g on basis of $0.05x10^{-8}$ mol per cm² at a concentration $20x10^{-8}$ mol per cc.

Table III. Desorption Measurements of Phosphate Sorbed on Galena

Galena	Phosphate Initially Sorbed Mol per g x 10⁸	Desorption Time	Mols Desorbed Mol per g x 10⁸	Concentration of Desorp. Soln. Mol per cc x 10⁸
2	363	14 min	0.53	0.02
	363	25 min	0.40	0.02
3	176; 604	1 hr	0.25	0.01
	176; 604	5 hr	0.75	0.03
	176; 604	15 hr	1.50	0.06

Table IV. Exchange of Phosphate Between Galena and Solution, 1 Min Exchange Time
($20x10^{-8}$ mol per cc Na_2CO_3)

Sorption Time, Min	Total Mols x 10⁸		Soln. Concentration Mol per cc x 10⁸	Fraction Exchanged
	In Soln.	On Galena		
5	356	179	17	0.11
13	328	215	16	0.12
80	220	349	8.6	0.17
120	146	442	5.0	0.27

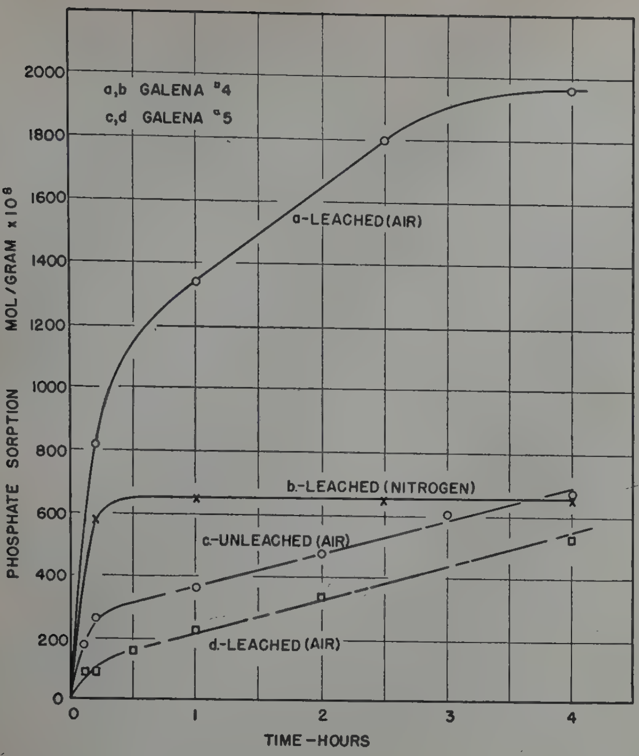

Fig. 2—Effect of air and of leaching on the rate of phosphate sorption by galena.

Curves a, b: Effect of air and of N_2 on sorption. Initial phosphate concentrations: a, $100x10^{-8}$; b, $50x10^{-8}$ mol per cc.

Curves c, d: Effect of leaching on sorption in air. Initial phosphate concentrations: c, $50x10^{-8}$; d, $49x10^{-8}$ mol per cc.

For comparison, the solubility of $Pb_3(PO_4)_2$ was measured in water and $20x10^{-8}$ mol per cc Na_2CO_3. In both media, the equilibrium phosphate concentration was $0.097 \pm 0.002x10^{-8}$ mol per cc.

Exchange: The extent of exchange in one minute between sorbed radioactive PO_4 on ground galena and inactive PO_4 in solution was studied. Any sorption which might have ocurred during the exchange measurement was negligible. The data as a function of time of sorption and the mols of phosphate in the solution and on the mineral are shown in Table IV.

For purposes of comparison, measurements were made of the exchange of PO_4 between solution and freshly precipitated $Pb_3(PO_4)_2$, Table V. These were necessarily at higher actual mols in precipitate.

Similar to the dithiophosphate case, exchange occurred here between phosphate ion and the freshly precipitated lead phosphate as well as with the sorbed phase on galena. Comparisons are again handicapped by the difficulties of attaining comparable conditions, so that definite conclusions cannot be drawn.

In the previous investigation on the interaction of dithiophosphate and galena, it was found that the heterogeneous, complex nature of the galena surface played a determining role in the rapid initial process. Because of this, it was natural that the rapid initial sorption of phosphate should possess points of similarity with that of dithiophosphate. Characteristic of both interactions were the following: completion in about 10 to 12 min, variability in the amounts taken up by the different galena preparations, decreased sorption on leaching, increased sorption on weathering, finite exchange rates, and slow desorption to produce low solution concentrations. From these similarities, it was expected that the phosphate-galena interaction should, as in the case of dithiophosphate, show characteristics of both chemisorption and chemical reaction.

Table V. Exchange of Phosphate Ion with Lead Orthophosphate

Soln. Concentration Mol per cc x 10^6	Total Mols Present x 10^6		Exchange Time Min	Fraction Exchanged
	In Soln.	In Precip.		
16.2	404	200	10	0.25
8.2	204	200	10	0.24
4.1	102	204	10	0.16
1.6	41	204	10	0.07

In line with this, it was possible to dismiss physical adsorption because of the finite exchange rate, and because of the failure, on desorption, to approach the same equilibrium concentrations existing during sorption. However, neither chemical reaction nor chemisorption could be excluded. On the one hand, chemisorption was plausible because on leached galena, for example, the amount taken up ($80x10^{-8}$ mol per g) was approximately that calculated ($90x10^{-8}$ mol per g) for close packing of PO_4 groups (18Å² from Fischer-Hirschfelder models) in a monolayer on a surface of 1000 cm² per g. This was also approached in the uptake by single crystals, namely an observed value of $0.06x10^{-8}$ mol per cm² as compared to the calculated of $0.09x10^{-8}$ mol per cm². On the other hand, chemical reaction leading to compound formation was indicated because the amounts taken up on unleached galena were equivalent to several monolayers, and furthermore, the concentration reached on desorption ($0.06x10^{-8}$ mol per cc) compared favorably with that expected from the solubility of lead orthophosphate. Evidently, therefore, the initial interaction with phos-

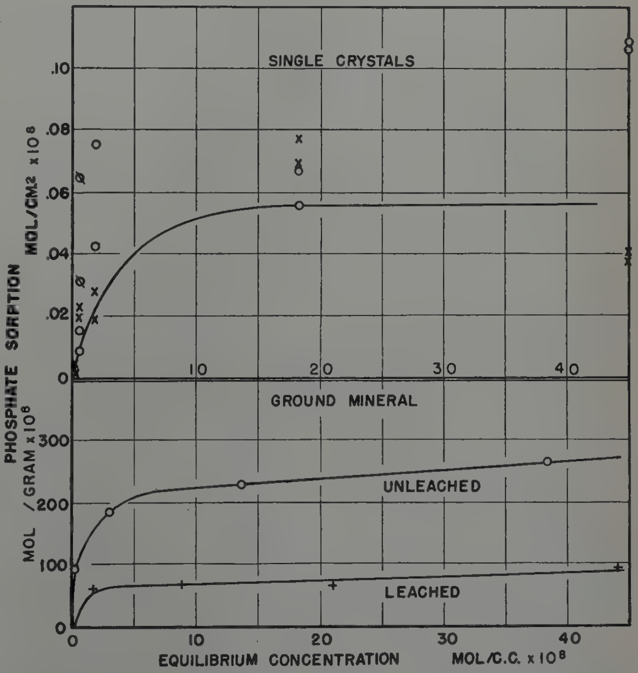

Fig. 3—Sorption isotherms (25°C) of phosphate on single crystals and on ground galena No. 5. Sorption time, 12 min.

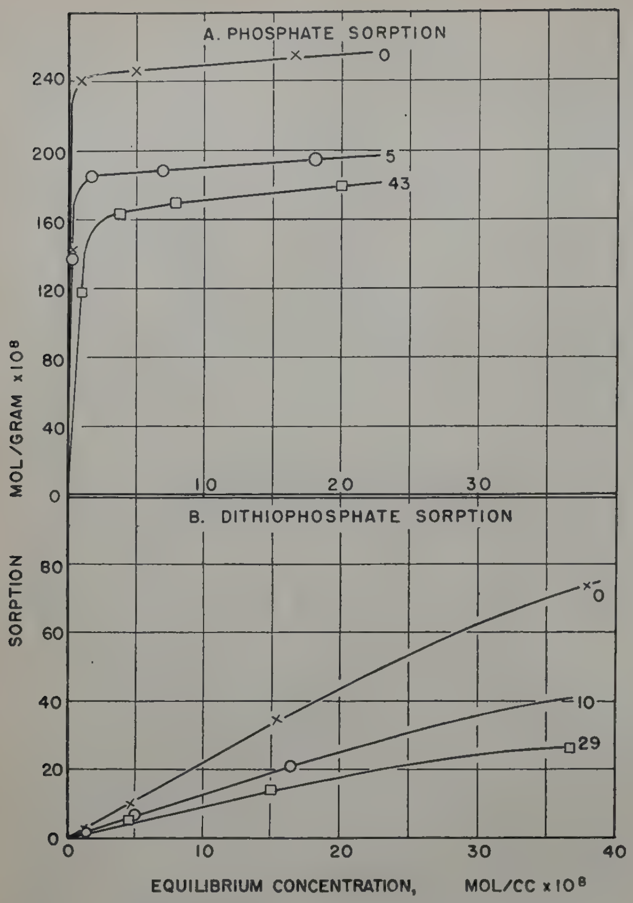

Fig. 4—Competition isotherms (25°C). Unleached galena No. 2; sorption time 12 min.

A. Phosphate sorption in the presence of dithiophosphate; initial dithiophosphate concentrations of 0, 5, and 43x10^{-8} mol per cc.

B. Dithiophosphate sorption in the presence of phosphate; initial phosphate concentration of 0, 10, and 29x10^{-8} mol per cc.

phate showed the same complexity observed with dithiophosphate.

Of considerable interest was the observation that no fixed ratio of phosphate to dithiophosphate could be found for the amounts taken up on the various galena preparations. Comparison of phosphate sorption, Table II, with that of dithiophosphate, Table II,[1] shows that the phosphate to dithiophosphate ratio varied from 1.0 to 0.33. This strongly suggests that if the interaction was a single process in one case, more than one process was involved in the other.

Table VI. Comparative Depression of Phosphate on Unleached and Leached Galena No. 5 by Dithiophosphate

Dithiophosphate Initial Conc. Mol per cc x 10⁸	Phosphate[a] Equil. Conc. Mol per cc x 10⁸	Phosphate[a] Sorption Mol per g x 10⁸	Depression, Pct
Unleached Galena			
0	29.5	210	0
22.5	30.6	188	10
75.0	32.9	142	32
150.	38.9	22	89
Leached Galena			
0	33.9	105	0
22.0	35.6	70	33
73.2	37.7	27	74
146.	38.6	9	91

[a] Initial Concentration Phosphate, 40x10^{-8} Mol per cc (20x10^{-8} Mol per cc Na_2CO_3)

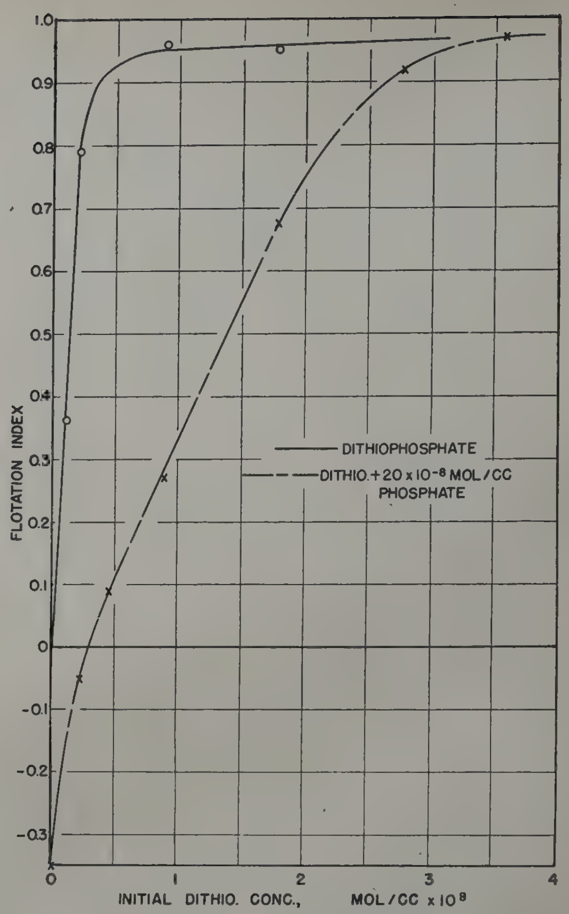

Fig. 5—Flotation of galena No. 2 by dithiophosphate, alone and in the presence of phosphate.

A feature of the phosphate interaction in air was the long, slow sorption following the rapid initial process. This was characterized by (1) dependence on dissolved oxygen in solution, (2) independence of rate on phosphate concentration (except at relatively low concentrations), and (3) equality of rates on leached and unleached mineral. These facts show that the slow process resulted from a primary reaction of oxygen in solution with lead sulphide to form complexes or compounds, which then reacted with phosphate. The phosphate eventually reached low concentrations (0.1 to 0.02x10^{-8} mol per cc) because the oxidation made lead continuously available. Since these final concentrations were comparable to the solubility of lead orthophosphate, it therefore appeared that the slow process resulted eventually in the formation of lead orthophosphate. The desorption also produced values of the same order of magnitude. Being the least soluble of the phosphates, the orthophosphate would be the eventual end product of the slow reaction.

Depression of Dithiophosphate by Phosphate

Sorption: The effect of phosphate on dithiophosphate sorption was studied by measurements of the 12-min sorption isotherms of each ion in the presence of the other. Such measurements could have been invalidated by exchange in solution of the tagged atom between the two ion species; actual determinations, however, proved that such exchange did not occur in the period of one hour over the concentration range employed.

Competition took place for an unleached galena surface as shown by the typical sorption data for galena No. 2 in Fig. 4 and for galena No. 5 in Tables VI and VII. Phosphate was a relatively stronger depressant than dithiophosphate. A point of difference existed between galenas No. 2 and No. 5 in that on No. 5 complete depression of either ion by the other was possible, while on No. 2 only partial depression could be obtained. This is further evidence of the variability and heterogeneity of the mineral surface. Evidently, on galena No. 2 a surface available to phosphate alone and a surface available to dithiophosphate alone must have been present in addition to a surface of competition.

The effect of leaching the mineral before sorption is illustrated in Tables VI and VII. After leaching, dithiophosphate was not as strongly depressed by phosphate and, conversely, phosphate was more strongly depressed by dithiophosphate. The degree of competition varied, therefore, with the composition of the mineral surface.

Flotation: In the presence of a relatively large concentration of phosphate, the flotation of unleached mineral was strongly depressed as illustrated by data on galena No. 2 in Fig. 5. The depression of the blank value, or weight floated in absence of collector, indicated that sorbed phosphate may have rendered the surface more wettable. Flotation increased with increasing collector concentration and asymptotically approached the maximum value at a concentration about eight times that when no phosphate was present.

Fig. 6 shows the effectiveness of small concentrations of phosphate in depressing flotation when the amount of collector sorbed was the minimum necessary for complete flotation. Measurements were made with a dithiophosphate concentration of 0.45x10^{-8} mol per cc corresponding to the knee of the normal flotation curve. On adding small quantities of phosphate, flotation was sharply depressed; reduction to one half the original value occurred at a phosphate equilibrium concentration of only 0.15x10^{-8} mol per cc.

Leached and unleached galena No. 5 showed a definite differences in flotation. For example, in a solution containing 2.5x10^{-8} mol per cc of dithiophosphate, the flotation of unleached galena was decreased 40 pct by the addition of 30x10^{-8} mol per cc phosphate, while that of leached galena appeared to be unaffected.

Discussion

The data on the competition of phosphate and dithiophosphate for a galena surface showed that the competition process was equally as complex as the interaction of the substances individually. In view of this, it was not surprising that the data were not amenable to quantitative treatment. By proper interpolation and arithmetical transformation, it was possible to calculate the equilibrium concentrations of the respective ions in solution for any given amount of dithiophosphate and of phosphate on the galena sample. These are the quantities necessary for any type of correlative calculation. The data of the interactions individually did not allow prediction of the relative amounts of each sorbed when in the presence of one another. Certainly, no prediction of the relative sorption from one galena preparation to another was possible. In no case could the entire competition picture be represented by isotherms derived on the assumption that two components were competing for one or for two surfaces. Finally, no satisfactory treatment of the data was obtained by attempts to apply solubility product considerations assuming coprecipitation of the two substances as lead salts. These points generally strengthen the prior conclusion that more than one process is involved in the interaction of these agents with galena. Also, they suggest that the quantitative aspects of competition in systems of this type are only learned by direct measurement. The value of radiotracer or other methods which directly provide quantitative knowledge of the amounts taken up and the resulting equilibrium concentrations cannot be over-emphasized, especially in view of the importance of competition in activator as well as depressant action.

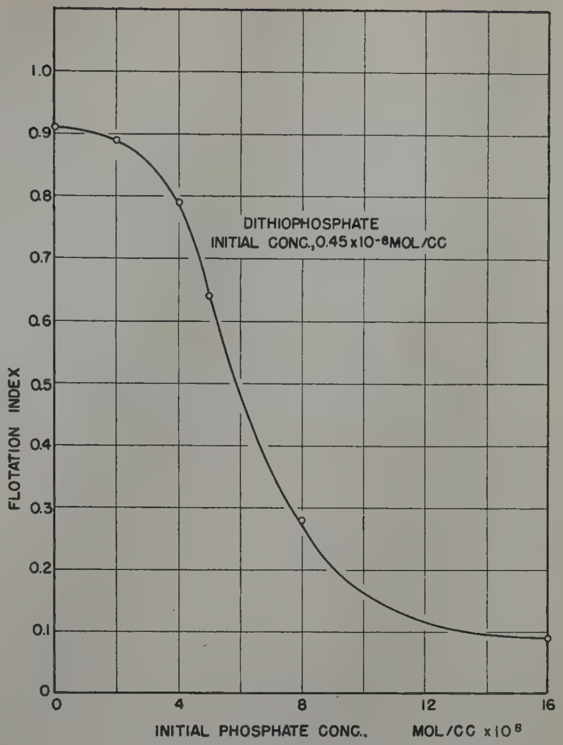

Fig. 6—Depression of flotation of galena No. 2 by phosphate. The initial dithiophosphate concentration was near the minimum necessary for complete flotation in the absence of phosphate.

Flotation of a mineral occurs when the amount of collector sorbed is sufficient, by virtue of its hydro-

Table VII. Comparative Depression of Dithiophosphate on Unleached and Leached Galena No. 5 by Phosphate

Phosphate Initial Conc. Mol per cc x 10^8	Dithiophosphate[a] Equil. Conc. Mol per cc x 10^8	Dithiophosphate[a] Sorption Mol per g x 10^8	Depression, Pct
Unleached Galena			
0	35.9	530	0
18.8	45.2	344	35
37.5	59.2	64	88
75.0	61.0	28	95
Leached Galena			
0	49.6	231	0
18.3	54.1	139	40
36.6	57.7	65	72
73.2	59.0	38	84

[a] Initial Dithiophosphate Concentration, 62.4x10^{-8} Mol per cc (20x10^{-8} Mol per cc Na_2CO_3)

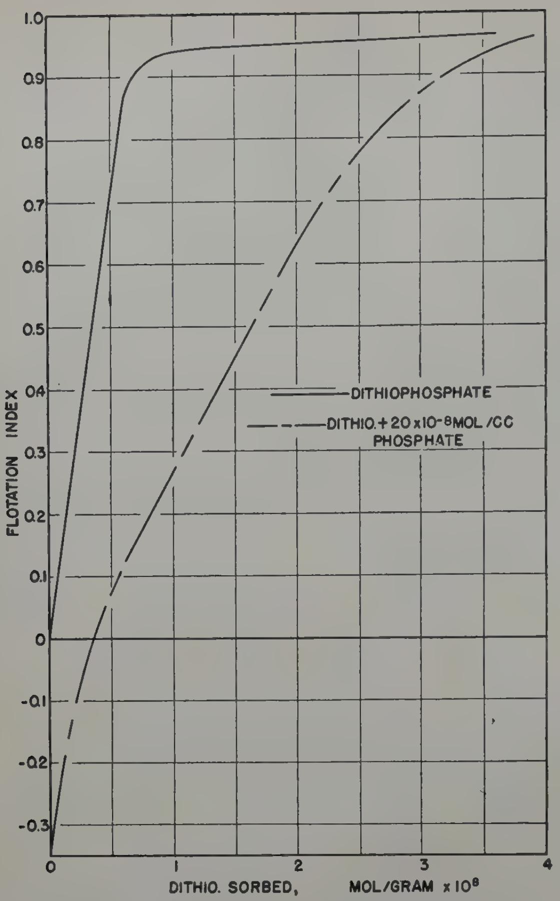

Fig. 7—Relation of the amount of dithiophosphate sorbed to the flotation for galena No. 2.

phobic groups, to maintain attachment of the mineral particles to the air-water interface. Flotation in solutions containing dithiophosphate and phosphate should accordingly be determined, not by the concentration of the dithiophosphate, but rather by the amount of dithiophosphate taken up, provided that the sorbed phosphate has no hydrophilic influence. A plot of flotation index versus dithiophosphate on the galena in the absence and in the presence of phosphate is shown in Fig. 7, as constructed from sorption and flotation data. It is clear that the amount of dithiophosphate sorbed is not solely controlling, for if such were the case the two curves would coincide. Evidently, the hydrophilic character of the sorbed phosphate has a significant influence on the effectiveness of a given sorbed quantity of dithiophosphate. This indicates that the characteristics of the depressing ion must be considered as well as the amount of collector taken up. Fig. 7 is also of interest as an example of the information which can be obtained by simple tracer measurements. As previously pointed out, complete flotation occurred with a relatively small amount of dithiophosphate on the surface when no phosphate was present (monolayer formation was estimated at 45×10^{-8} mol per g of collector). The strong depression by phosphate at concentrations corresponding to the knee of the dithiophosphate curve was obviously the result of the critical dependence of flotation on the amount of collector sorbed in this region. The mineral surface was also a determining factor, since leaching increased the relative amount of dithiophosphate to phosphate taken up and correspondingly increased flotation.

All of the considerations of this and the first paper on the interaction of dithiophosphate with galena have been mainly focused on the initial rapid process, since it appears of primary interest in flotation. The complex nature of the interaction has been indicated. For additional clarification, investigation at the lowest possible concentrations and surface coverages would be valuable since only small amounts are necessary to produce maximum flotation. In addition, measurements at several temperatures should prove helpful because they would permit estimates of the activation energies and the heats of the processes involved.

Summary

A previous study[1] of the interaction of dithiophosphate with galena was extended by a study of the depressant action of phosphate on this system. The depression was investigated by correlative measurements of the competition of each ion for galena in sorption and in flotation. Using selective radiotagging methods, measurements of one ion in the presence of the other could be made as simply as if only one species had been present. It was necessary for the interpretation of these data to study also the interaction of phosphate alone with galena.

The interaction of phosphate with galena possessed considerable similarity to that of dithiophosphate. Characteristics of both chemisorption and surface chemical reaction were evident. The complexity of the interaction resulted largely from the heterogeneous nature of the galena surface.

Competition in sorption between dithiophosphate and phosphate was equally as complex as the sorption process of the individual ions. Although qualitatively the results were those expected from the observed behavior of the separate ions, no satisfactory quantitative treatment of the data could be made. This strengthened the conclusion that more than one process was involved in the interaction of these agents with galena, and suggested that the quantitative aspects of such systems may only be determined by direct measurements.

The flotation data generally confirmed the hypothesis that the depressant action of phosphate was a result of reducing the amount of dithiophosphate taken up by the mineral. The hydrophilic character of the sorbed phosphate, however, appeared to influence the effectiveness of the sorbed collector for flotation. Leaching of the mineral reduced the effectiveness of the depressant by increasing the relative amount of collector sorbed, thus emphasizing again the dominant role of the mineral surface in flotation processes.

Acknowledgment

We wish to express our appreciation for encouragement and support from S. J. Swainson and R. B. Barnes, for helpful discussion with R. B. Booth, S. A. Falconer, J. E. Carpenter, and for technical assistance from various other members of these Laboratories during this investigation.

Reference

[1] G. L. Simard, J. Chupak, and D. J. Salley: Radiotracer Studies on the Interaction of Dithiophosphate with Galena. *Transactions* AIME (1950) **187**, 359-364; *Mining Technology* (March 1950) TP 2815.

Geology, Mining, And Uses Of Strategic Pegmatites

by Richard H. Jahns

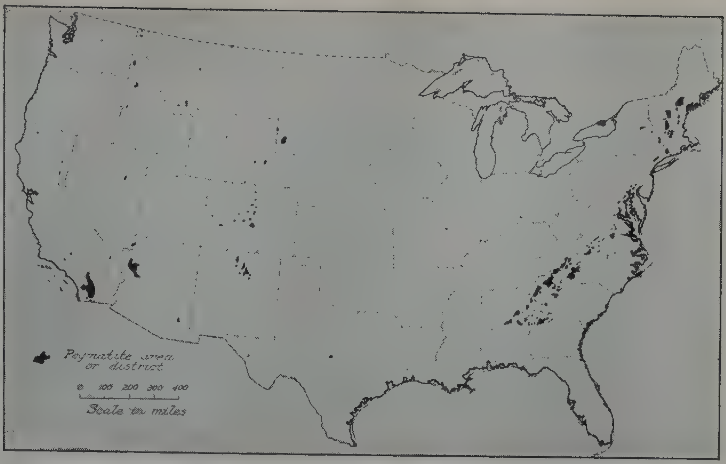

Fig. 1—Principal pegmatite areas and districts of the United States.

Such minerals as beryl, lepidolite, sheet muscovite, spodumene, and tantalite-columbite are obtained chiefly from pegmatite bodies that are internally zoned. As shown by examples of such pegmatites from the Southwestern and Southeastern States during World War II, studies of their internal and external structure have value in prospecting and the planning of exploration, development, and mining.

GRANITIC pegmatite deposits are the chief source of commercial feldspar, sheet mica, beryllium, tantalum-columbium, and lithium minerals, and certain types of kaolin. They also have yielded significant quantities of cassiterite, gems, scrap mica, molybdenite, tungsten minerals, uranium-thorium and rare-earth minerals, and zircon, either directly or as the sources of placer deposits. The output from pegmatite mines in the United States is small as compared to other mineral products in terms of bulk or value, and much of it comprises minor metals and nonmetals. Nevertheless, pegmatite minerals play a vital part in domestic industrial economy, particularly in the ceramic and electrical industries. Numerous special purpose uses also are important, even though they require small quantities of raw materials.

Never was United States dependence upon pegmatite mining more clearly emphasized than during World War II, when greatly expanded demands and uncertainties of foreign sources of supply caused much concern. Such commodities as beryl, tantalite, and sheet muscovite remained high on the critical list for months at a time, and for some minerals there was a real struggle to achieve and maintain a favorable ratio of supply to demand. Satisfactory stock piles of certain sizes and qualities of sheet mica, for example, were not built up until late in 1944, when wartime requirements already had begun to level off or even to decrease. Production of all the strategic pegmatite minerals from domestic deposits during World War II constituted only 10 pct or less of the total domestic consumption of these minerals, but the importance of this contribution should not be minimized. Domestic production often represented the difference between increase and further reduction of existing stocks of commodities already in seriously short supply, and in the early stages of the war it was the only assured source if imports were cut off.

During the recent war period, the U. S. Geological Survey carried on a program of pegmatite studies in the New England and Southeastern States, in the Black Hills area of South Dakota, in the Rocky Mountain region of Wyoming, Colorado, and New Mexico, and in various parts of Washington, Idaho, Utah, Nevada, Arizona, and California as shown in Fig. 1. Several papers and reports already have appeared as an outgrowth of this work,[1-12] and others are in preparation. This paper outlines some results of data obtained by the writer during the period 1942 to 1946, when he was engaged in studies of pegmatites in the Southeastern, Southwestern, and Rocky Mountain States.

Beryllium Minerals

Beryl is the present commercial source of beryllium metal and beryllium compounds, which are used in ceramics, in the preparation of X-ray tubes and fluorescent lamps and screens, in special processes of paint and textile manufacture, and in the optical systems of some electrical instruments. The metal also is used in nuclear physics, chiefly as a source of neutrons. Beryllium is alloyed with aluminum for certain light-metal uses and is a constituent of some nickel and iron alloys. Currently the chief demand, however, is for copper-base alloys, which are exceptionally resistant to fatigue and wear, are responsive to hardening treatments after

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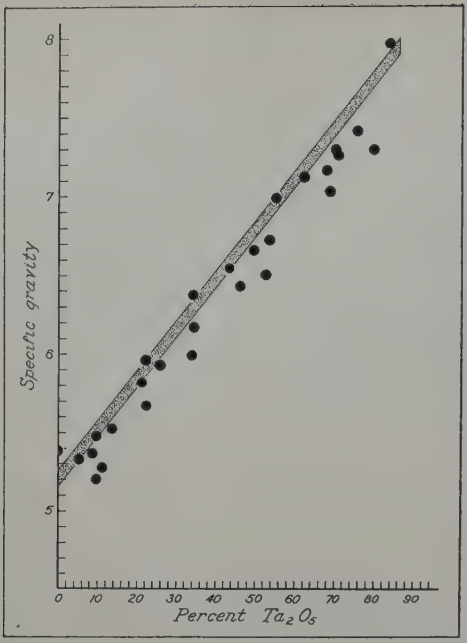


Fig. 2—Relation between specific gravity and tantalum oxide content in the columbite-tantalite series.

Stippled area represents theoretical range of values based on assumption that specific gravity of pure columbite is 5.20±0.05 and that of pure tantalite is 7.95±0.05. Black circles represent published analyses.

being worked soft, and are harder and otherwise superior to copper in structural characteristics. In addition, they are good electrical conductors, nonmagnetic, and nonsparking. Alloys of the beryllium-copper group are used, for example, in nonsparking tools and in springs, contact plates, bushings, shims, and corrosion-resistant parts of motors and gages. They also are employed for parts in precision instruments and machines.

Prices for clean-cobbed domestic beryl at the mine were quoted at $30 to $35 per short ton for several years prior to 1941, but they gradually rose during the succeeding wartime months until by May 1943 a price of $12 per dry short-ton unit was guaranteed by the Metals Reserve Co. for ore containing a minimum of 8 pct beryllium oxide (BeO). In June 1944, the price was raised to $14.50 per unit, but federal purchases were discontinued on Jan. 1, 1945. Subsequent open market transactions were carried on at distinctly lower price levels for a time, but prices since have risen to points well above the previous wartime high.

The BeO content of pure beryl varies with the proportion of certain alkalies present, notably sodium and cesium and, in general, ranges from less than 10 pct to a theoretical maximum of about 14 pct. Although satisfactory analytical methods have been developed for the determination of beryllium in ores,[13] they are time-consuming and expensive. Accurate analysis of low-grade material is particularly difficult. A fairly rapid qualitative method for recognition of BeO in minerals has been devised[14] and is reliable for material containing 1 pct or more of the oxide. An improved photometric method of analysis also has been reported,[15] and spectrochemical analysis, which is rapid and only moderately expensive, has been used successfully for numerous BeO determinations, especially in low-grade ores.[16] Dr. W. T. Schaller of the Geological Survey has been investigating the relations between indices of refraction, which are easily determined, and the BeO content of beryl, and this work promises to yield a rapid and inexpensive means for approximate analysis.

Table I. Composition of Principal Beryllium Minerals[a]

Mineral	Simplified Formula	Theoretical Content of BeO, Pct
Beryl	$Be_3Al_2(SiO_3)_6$	13.97
Phenakite	Be_2SiO_4	45.45
Chrysoberyl	$BeAl_2O_4$	19.71
Helvite[b]	$R_4Be_3Si_3O_{12}S$	12.58 to 13.52
Gadolinite	$Be_2FeY_2Si_2O_{10}$	10.69
Beryllonite	$NaBePO_4$	19.69

[a] Includes minerals of present and potential commercial importance.

[b] Mineral group, comprising three theoretical end members: Helvite, $Mn_4Be_3Si_3O_{12}S$; Danalite, $Fe_4Be_3Si_3O_{12}S$; Genthelvite, $Zn_4Be_3Si_3O_{12}S$.

The beryl produced in the southwestern and southeastern United States is a byproduct derived from deposits of feldspar, lithium minerals, mica, or other pegmatite constituents, and hence the general output ordinarily varies according to market conditions for these other minerals. The beryl is recovered from the host pegmatite by rough cobbing and hand sorting, so only deposits containing coarse crystals and masses are of current economic interest. Several laboratory investigations, however, have demonstrated that at least some pegmatite containing small and scattered masses of beryl can be milled with reasonably good recovery of the mineral in the form of marketable concentrates.[17, 18]

The Black Hills area of South Dakota is the leading source of domestic beryl, and small quantities are obtained at irregular intervals from deposits in at least 14 additional states. The bulk of the beryl consumed in the United States, however, is imported from South American and other countries.

Other beryllium-bearing minerals, such as chrysoberyl, gadolinite, and phenakite, occur in many pegmatites, both in this country and abroad, but no deposits of present commercial importance are known. Phenakite, with a BeO content of about 45 pct, is a

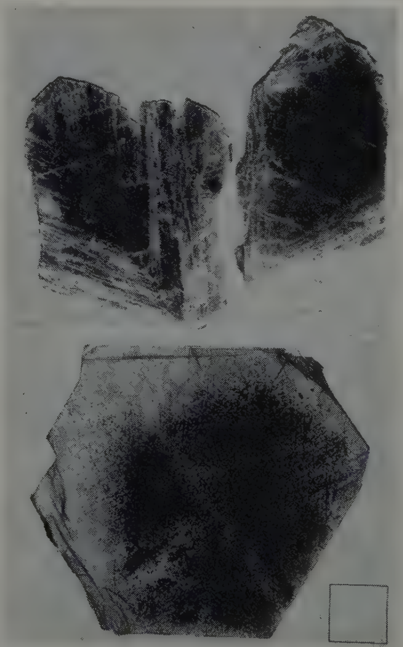


Fig. 3—Cleavage pieces of muscovite.

Top—Pieces marred by "A" reeves and cracks. Bottom—Flat piece with abundant hematite inclusions in form of flattened trigonal lattices. (One-inch square shows scale.)

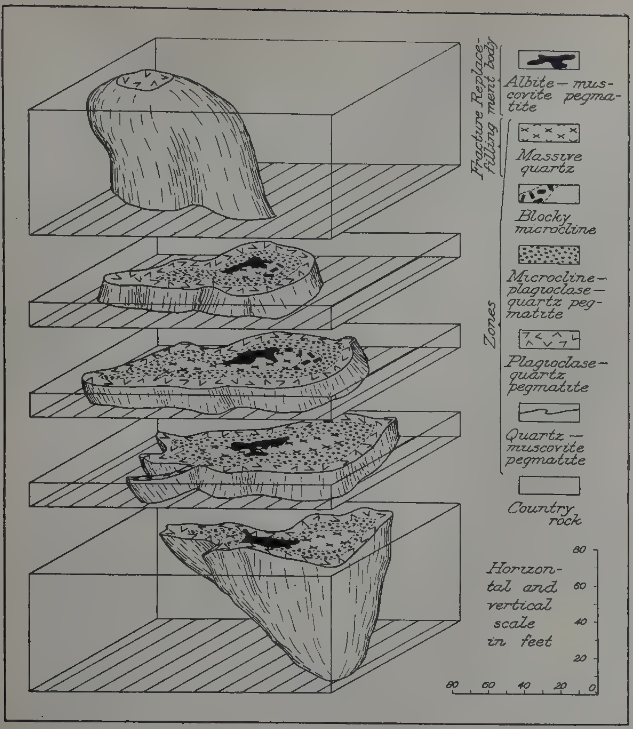

Fracture filling
Replacement body
Albite–muscovite pegmatite
Massive quartz
Blocky microcline
Microcline–plagioclase–quartz pegmatite
Zones
Plagioclase–quartz pegmatite
Quartz–muscovite pegmatite
Country rock
Horizontal and vertical scale in feet
80
60
40
20
0

Fig. 4—Isometric block diagram of a typical steeply plunging podlike pegmatite body showing distribution of zones and other units at different levels.

locally abundant constituent of the Morefield pegmatite, Amelia County, Va. A reserve of several tons of this mineral might well be present, but most of the phenakite masses are too small to be recovered by ordinary hand sorting. Helvite and danalite, complex beryllium-bearing silicate minerals[19] occur abundantly in the Iron Mountain district, N. Mex. The deposits are of contact metamorphic rather than pegmatitic origin. Most are low in grade, and the difficulties involved in their metallurgical treatment appear to preclude their successful competition with beryl ores as a current source of beryllium.[20] The composition of principal beryllium minerals is given in Table I.

Lithium Minerals

Salts of lithium are used in pharmaceuticals, storage batteries, flares and fireworks, fluxes and special lubricants, as well as in the curing of meat, manufacturing of textiles and ceramics, refining of metals, smelting of iron ore, purifying of helium,[21] and dehumidifying of air in air-conditioning equipment. Lithium hydride has been employed as an effective transporter of hydrogen in self-inflating rafts, balloons, and other devices, and the metal is a minor constituent of some special-purpose alloys. The chief raw materials used for the manufacture of lithium compounds are the pegmatite mineral spodumene and the lithium-bearing brines of several saline lake deposits. Other pegmatite source minerals include amblygonite, zinnwaldite, and triphylite, but neither these nor spodumene contain large proportions of lithium or lithium oxide, as shown in Table II.

Spodumene can be employed directly in the manufacture of glass to neutralize shrinkage during cooling and also is an ingredient of some ceramic mixes. Lepidolite, the lithia mica, is much more widely used in glass making than as a source of lithium salts and is commercially as important as spodumene. Not only is it an excellent fluxing material, but it increases the luster, weather resistance, and strength characteristics of glasses, while reducing their coefficients of expansion. Such glasses are in great demand for high-pressure gages, electronic tubes, and other devices subject to mechanical stresses or sudden temperature changes. Lepidolite also is an ingredient of many high-quality porcelains and enamels in which it is an effective opacifier.

The average price quoted for spodumene during recent years has been about $30 per short ton of clean cobbed or milled material, generally with a minimum acceptability limit of 6 pct contained Li_2O. Prices for amblygonite, which contains a higher proportion of lithium, ordinarily have ranged from $40 to $50 per short ton. Lepidolite containing 3 pct Li_2O generally has commanded a price of $15 to $27 per short ton at the mine. Spodumene was allocated to consumers during the wartime period Dec. 5, 1942, to Sept. 30, 1944, when foreign supply was uncertain and domestic production facilities were not adequate in terms of demand. During this time prices rose to $50 or more per ton.

Most domestic spodumene has been obtained from deposits in North Carolina, South Dakota, and New Mexico. A group of pegmatites in the Kings Mountain area of the Carolina Piedmont constitutes by far the greatest domestic reserve of the mineral. This reserve may well amount to several million tons. Amblygonite is mined from time to time in

Table II. Composition of Commercial Lithium Minerals from Pegmatites

Mineral	Simplified Formula	Theoretical Content of Li_2O, Pct	Proportion of Li_2O in Most Marketed Concentrates, Pct
Spodumene	$LiAlSi_2O_6$	8.03	>6.0[a]
Lepidolite	$KLi_2AlSi_4O_{10}F_2$[b]	7.65[b]	>2.5
Amblygonite	$LiAlFPO_4$	10.10	>8.0
Zinnwaldite	$LiKFeAl_2F_2Si_3O_{10}$	3.40	>1.5
Triphylite	$LiFePO_4$	9.47	>3.5
Lithiophilite	$LiMnPO_4$	9.52	
Petalite	$LiAlSi_4O_{10}$	4.89	>3.0

[a] Concentrates containing as little as 4.0 pct Li_2O are sometimes marketed.
[b] For polylithionite end member.

Table III. Composition of Principal Tantalum-Columbium Minerals[a]

Mineral	Simplified Formula	Theoretical Content of Ta_2O_5 or Cb_2O_5, Pct	Proportion of Ta_2O_5 or Cb_2O_5 in High-Grade Concentrates, Pct
Tantalite[b]	$(Fe,Mn)Ta_2O_6$	86.12 Ta_2O_5[c]	>70 Ta_2O_5
Columbite[b]	$(Fe,Mn)Cb_2O_6$	78.88 Cb_2O_5[c]	>65 Cb_2O_5
Microlite[b]	$(Na,Ca)_2Ta_2O_6(O,OH,F)$	>82 Ta_2O_5	>70 Ta_2O_5
Pyrochlore[b]	$NaCaCb_2O_6F$	73.05 Cb_2O_5	>60 Cb_2O_5
Hatchettolite	$(Na,Ca,U)_2(Ta,Cb)_2O_6(O,OH,F)$	>65 $Ta_2O_5+Cb_2O_5$	>60 $Ta_2O_5+Cb_2O_5$
Samarskite	$(Y,Er,Ce,U,Ca,Fe,Th)(Cb,Ta)_2O_6$	>55 $Cb_2O_5+Ta_2O_5$	>45 $Cb_2O_5+Ta_2O_5$
Tapiolite[d]	$(Fe,Mn)(Ta,Cb)_2O_6$	86.01 Ta_2O_5	>70 Ta_2O_5
Fergusonite[b]	$(Y,Er,Ce,Fe)CbO_4$	>54 Cb_2O_5	>50 $Cb_2O_5+Ta_2O_5$
Formanite[b]	$(Y,Er,Ce,Fe)TaO_4$	>66 Ta_2O_5	
Euxenite[e]	$(Y,Ca,Ce,U,Th)(Cb,Ta,Ti)_2O_6$	>55 $Cb_2O_5+Ta_2O_5$	

[a] Includes minerals of present and potential commercial importance.
[b] End member of a tantalian-columbian mineral series.
[c] For material with Fe:Mn ratio of 1:1.
[d] Tantalum-columbium mineral series that include high-tantalum members but probably no corresponding high-columbium members.
[e] General name for members of a columbium-tantalum-titanium mineral series that are relatively rich in Ta and Cb.

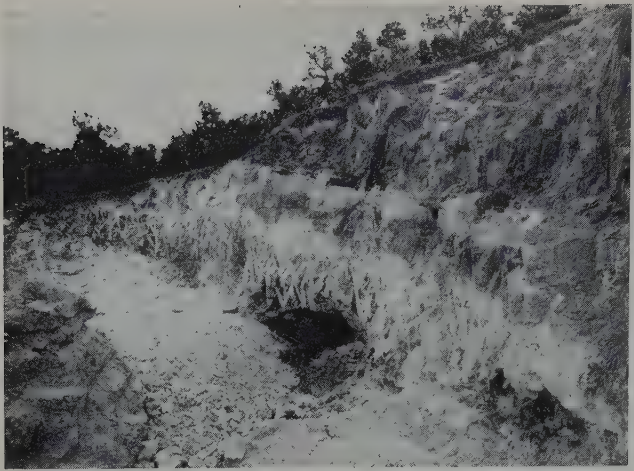


Fig. 5—Harding quarry, Taos County, N. Mex. This view of the south wall shows the upper half of the gently dipping pegmatite dike. Note the well-defined layering.

South Dakota, and lepidolite has been recovered recently from deposits in South Dakota, Colorado, and New Mexico. Both minerals were obtained from the Stewart mine in southern California prior to 1926, and production of lepidolite from this mine was large.

Some lithium minerals are imported from Southwest Africa, Argentina, Brazil, Sweden, central and southwestern Europe, Australia, and Canada, but during most years the consumption of such materials in this country does not greatly exceed domestic production.

Recovery of lithium minerals from the host pegmatite ordinarily is a simple matter, involving hand cobbing and picking. Many crystals or crystalline aggregates of spodumene, lepidolite, or amblygonite are so large that they can be broken and shipped without sorting. In other places these minerals are intimately intergrown and can be marketed as mixed lithium-bearing material. Lithium minerals have been recovered by milling in at least three places during recent years. The flotation plant of the Solvay Process Co. near Kings Mountain, N. C., yielded commercial spodumene concentrates during the period 1943 to 1945.

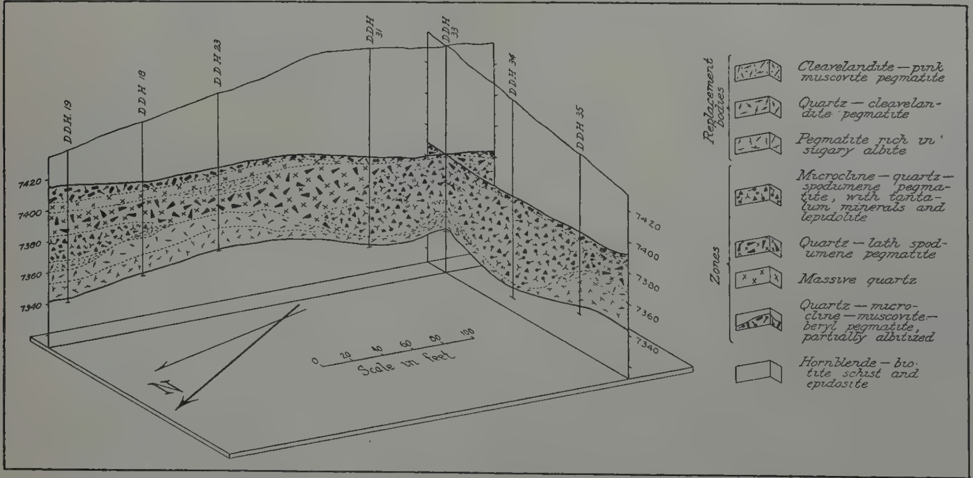
Tantalum-Columbium Minerals

The metals tantalum and columbium are derived mainly from members of the tantalite-columbite series and to an appreciable extent from microlite and other minerals, see Table III. Columbium is alloyed with nickel, copper, and aluminum. Columbium-bearing alloy steels, with their favorable welding characteristics and high-temperature strength properties, are in demand for turbine and aircraft engine parts. Tantalum metal is used in radio and neon tubes, where its gas absorption properties are important, and its chemical inertness makes it unusually suitable for instruments and equipment that are exposed to corrosive liquids and fumes. It is employed in the manufacture of synthetic rubber and is uniquely satisfactory as a surgical metal. Tantalum is alloyed with columbium and tungsten to form dies and cutting tools, which also are made with cemented tantalum carbides. Tantalum-bearing glass is used in special camera lenses and other optical equipment.

The most desirable ores are microlite, low-tantalum columbite, and low-columbium tantalite that contain little tin or titanium. Columbium ore was valued at \$0.22 to \$0.40 per lb, and tantalum ore at \$1.80 to \$2.75 per lb in the years preceding World War II, but prices rose much higher as a result of wartime demands after 1941. The highest prices paid by the Metals Reserve Co. were in effect during the period July 1943 to January 1945, after which most purchases were discontinued. They ranged from \$2.20 per lb of contained tantalum oxide for concentrates assaying 40 pct Ta_2O_5 to \$4.65 for 75 pct concentrates, see Table IV. Columbite containing 50 pct Cb_2O_5 or more was purchased at \$0.50 per lb of contained oxide.

Accurate determination of grade is accomplished by chemical analysis, a slow and costly process, but rough estimates for members of the tantalite-columbite series can be made on the basis of the consistent relation between specific gravity and Ta_2O_5 content. In general, the specific gravity rises with increase in proportion of tantalum oxide as shown in Fig. 2. The specific gravity of most columbite ranges from 5.2 to 6.5 and that of all material commercially marketable as tantalite is greater than 6.5. Determinations can be made easily and quickly by means of gravity balances, which are available in many laboratories. Preliminary investigations of microlite from several deposits in the United States suggest that a similar relation holds for varieties of this mineral, with a general rise in specific gravity from 5.5 for material containing less than 60 pct Ta_2O_5 to as much as 7.0 for very high grade material.

Tantalum-columbium minerals occur in many pegmatite districts of the United States, but the



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DDH 18
DDH 23
DDH 31
DDH 33
DDH 34
DDH 35
7420
7400
7380
7360
7340
7420
7400
7380
7360
7340
N
0
20
40
60
80
100
Scale in feet
Replacement bodies
Clearelandite—pink muscovite pegmatite
Quartz—clearelandite pegmatite
Pegmatite rich in sugary albite
Zones
Microcline—quartz—spodumene pegmatite, with tantalum minerals and lepidolite
Quartz—lath spodumene pegmatite
Massive quartz
Quartz—microcline—muscovite—beryl pegmatite, partially albitized
Hornblende—biotite schist and epidosite

Fig. 6—Isometric plate diagram of a part of the Harding pegmatite, Taos County, N. Mex. See Fig. 9 for locations of diamond-drill holes.

average annual output has amounted to little more than 6000 lb during the past 30 years. Crystals and irregular chunks of tantalite and columbite are recovered from time to time as hand-cobbed byproducts from operations for feldspar and other pegmatite minerals, but very few pegmatites have been mined for tantalum-columbium minerals alone. Notable exceptions are the Harding pegmatites of northern New Mexico, which during World War II yielded the bulk of domestic tantalum production in the form of microlite concentrates. Additional microlite has been obtained since 1942 as a byproduct from other pegmatites in New Mexico, Colorado, and South Dakota.

Nearly all tantalum-columbium ore consumed in the United States is tantalite-columbite obtained from deposits in other countries, mainly Africa, South America, and Australia. Much of the highest-grade tantalite occurs in Australia and parts of South America. During recent years some tantalum was brought into this country in the form of smelter slags derived from central African tin ores, but this material contained very low percentages of Ta_2O_5. Other tantalum-bearing minerals such as microlite, hatchettolite, samarskite, and fergusonite, are widespread in their occurrence but are so sparsely distributed in most tantalum-bearing pegmatites that they ordinarily are of little economic interest.

Sheet Muscovite

The uses of sheet muscovite are based upon its perfect cleavage, remarkably low conductivity of heat and electricity, high dielectric strength, noninflammability, mechanical strength, flexibility, elasticity, transparency, luster, and the ease with which it can be worked into final form. The degrees of emphasis placed upon given properties by purchasers depend upon the specific end uses involved.[22–24] Flexibility is particularly important, for example, in the "cigarette" mica used in spark plugs for aircraft engines. This material, in films twelve ten-thousandths of an inch or less thick, is wrapped around rodlike spindles a little more than ⅛ in. in diam.[24] Condenser mica, in contrast, is valued because of its dielectric properties, and the use of mica for windows in furnace walls and doors is founded upon its transparency, heat resistance, and mechanical strength.

Table IV. Price Schedule for Domestic Tantalite Concentrates, July 1943 to January 1945[a]

Proportion of Ta_2O_5 in Ore, Pct[b]	Price per Lb of Contained Ta_2O_5 $	Proportion of Ta_2O_5 in Ore, Pct[b]	Price per Lb of Contained Ta_2O_5 $	Proportion of Ta_2O_5 in Ore, Pct[b]	Price per Lb of Contained Ta_2O_5 $
40	2.20	52	3.04	64	3.88
41	2.27	53	3.11	65	3.95
42	2.34	54	3.18	66	4.02
43	2.41	55	3.25	67	4.09
44	2.48	56	3.32	68	4.16
45	2.55	57	3.39	69	4.23
46	2.62	58	3.46	70	4.30
47	2.69	59	3.53	71	4.37
48	2.76	60	3.60	72	4.44
49	2.83	61	3.67	73	4.51
50	2.90	62	3.74	74	4.58
51	2.97	63	3.81	75	4.65

[a] Purchases made by Metals Reserve Co. at designated depots; schedule applies to lots in excess of 200 lb dry weight.
[b] Minimum acceptable Ta_2O_5 content is 40 pct; maximum allowable contents of SnO_2 and TiO_2 are 3 pct.

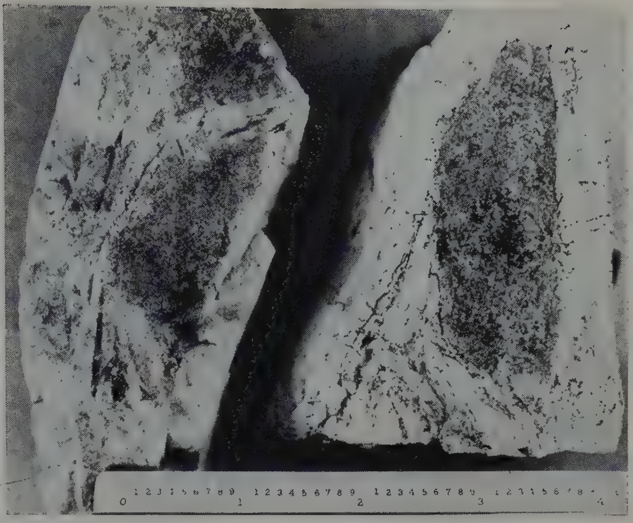

Fig. 7—Specimens of tantalum-lithium ore from the Harding pegmatite, Taos County, N. Mex.

High-grade ore. Aggregates of microlite, lithia mica, and minor tantalite are enclosed by laths of spodumene. Dark strips in specimen at left are smoky quartz.

A very high proportion of sheet mica is used as an electrical insulating material. Washers, disks, and other small trimmed or stamped forms are not only employed as such, but they can be built up into rods, tubes, or other articles by cementing them with shellac, glyptol, or a similar bonding medium. Simple and composite pieces are used, for example, as sleeves, studs, tubes, washers, bushings, laminations, and thin perforated plates in condensers, transformers, small heating elements, rheostats, fuses, incandescent bulbs, radio and electronic tubes, various types of coils, and in acoustic, X-ray, and other specialized equipment. Thin splittings are built up into mica board or applied as facing on paper, cloth, and other materials used in the manufacture of heater elements; commutators; boards, panels, and other mounting forms; parts of condensers; and many other electrical devices.

The properties most desired in sheet muscovite of superior quality are flatness, uniform splitting characteristics, reasonable hardness and flexibility, elasticity, transparency, freedom from inclusions of other minerals, and freedom from such structural imperfections as cracks, tears, pinholes, warps, and ripples, see Fig. 3. The first rough separation of mica generally is made at the mine, where obvious scrap material is separated from the better books. Adhering fragments of quartz, feldspar, and other foreign material are then cobbed away from those books that contain sheet mica of usable quality. Some of this rough cobbed or selected mine-run mica is sold as such to jobbers or manufacturers, but at most mines it is prepared further. The books are split or "rifted" by means of knives, generally into plates 3/16 in. or less in thickness. Most defective laminae removed during this rifting process are discarded as shop scrap, which is distinctly less impure than typical mine scrap.

After rifting, the ragged and broken edges of many plates are removed with the fingers, a process known as thumb trimming. This is an especially common practice in districts where much of the mica is marred by structural imperfections. Some thumb-trimmed material is sold to manufacturers as such, but most is further trimmed with knives or shears and its value thereby increased. During recent years attempts have been made in the United States to employ several forms of blades and saws for semimechanized mica trimming, but without much success.

A large proportion of sheet mica is consumed in the form of splittings. These are films 0.0007 to 0.001 in. thick that generally are cleaved from the smaller sizes of sheet stock. Some also are derived from thin films or skimmings that are accumulated during the rifting of larger sheet material. Splittings are used in the manufacture of built-up mica board and other forms of electrical insulation. Although many mechanical devices have been tested for their preparation, most splittings are still made outside the United States by hand methods, generally in places where labor costs are very low.

The cut mica blocks that represent sheet material are further processed into disks, washers, and thin plates of various sizes and shapes. This processing generally involves additional splitting, followed by trimming, cutting, punching, or stamping into more or less standardized patterns. Most of this material is then cut to final form, if necessary, by the manufacturers of the devices in which the mica is to be used. Composite forms can be built up to any desired thickness by the cementing together of individual pieces.

In general, only a small proportion of the rifted and trimmed sheet mica is represented in the finished product. The bulk of such material is trimmed or cut away as waste, which is then marketed as scrap of superior grade.

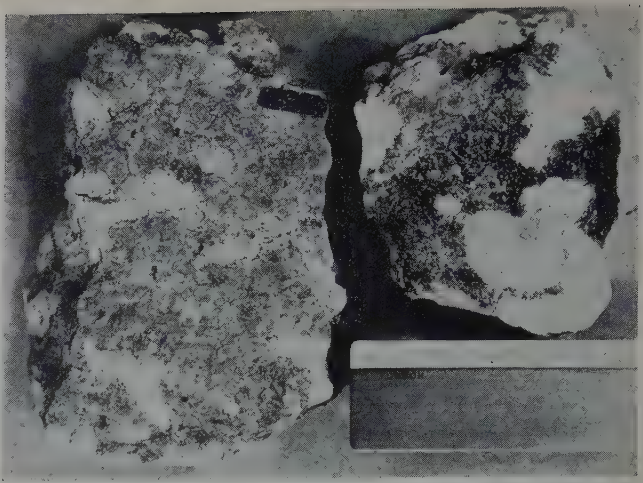

Fig. 8—Specimens of tantalum-lithium ore from the Harding pegmatite, Taos County, N. Mex.

"Spotted rock." Irregular masses of spodumene (white) are enclosed by an aggregate of microcline, albite, and minor tantalum minerals in specimen at left, and by purple lepidolite in specimen at right.

Heavy demands for sheet mica of good quality are made during modern wartime periods. Such mica is used, for example, as splittings in the form of built-up mica commutator segments and coil insulations for motors and generators, and in transformers, switchboards, blasting apparatus, and aircraft generators and sparkplugs.[25, 26] In addition, an unprecedented problem was caused during World War II by the demands and exacting requirements for condenser mica. Developments in the field of military radio and electronic equipment focused attention on mica condensers, owing to the constancy and excellency of their electrical properties under varying physical conditions.[26]

Some of the problems encountered in increasing the domestic supply of so-called strategic sheet mica have been described and discussed in recent literature.[1, 23, 26–28] To stimulate production of such mica, the Metals Reserve Co., a subsidiary of the Reconstruction Finance Corp., designated the Colonial Mica Corp. as its agent, with authority to purchase mica of certain types and to assist the operators of mines with equipment leases, development loans, and consulting services on problems of mica mining and preparation. A market for mica of superior quality was assured at favorable prices and for specified periods.

Table V. Price Ranges of Clear and Stained Sheet Mica in the Southeastern States During the Period 1910-1940

Size, In.	Range in Price, $ per Lb Clear	Stained
1½x2	0.12 - 0.60	[a]
2x2	0.22 - 1.05	0.06 - 0.40
2x3	0.38 - 1.45	0.10 - 0.60
3x3	0.58 - 2.00	0.15 - 1.25
3x4	0.78 - 2.30	0.30 - 1.50
3x5	0.95 - 2.70	0.48 - 1.75
4x6	1.75 - 3.65	0.70 - 2.25
6x8	2.25 - 7.25	1.25 - 2.50
8x10	3.50 - 11.50	2.00 - 3.00

[a] Under ordinary conditions the smallest size of stained mica purchased as sheet material in the Southeastern States is 2x2 in.

Table VI. Price Schedules for Domestic Clear Sheet Mica During Period December 1941-February 1945[a]

Size, In.	Prices, $ per Lb Private Purchasers April-May 1942	Colonial Mica Corp. June 1942[b]	November 1942[b]	May 1943[c]	February 1944[c]	August 1944		Private Purchasers February 1945[f]
Punch	0.12 - 0.16	0.22	0.30	5.00	6.00	6.00[d]	2.25[d]	0.08 - 0.15
1½x2	0.50 - 0.65	1.10	2.40					1.00
2x2	0.95 - 1.10	1.75	3.52			6.00[e] and 8.00[d]		1.40
2x3	1.50 - 1.85	2.75	4.64					2.00
3x3	2.00 - 2.35	3.50	5.12					2.55
3x4	2.25 - 2.60	4.25	6.08					3.00
3x5	2.75 - 3.00	5.00	7.04					3.45
4x6	3.75 - 4.00	6.25	8.00					4.30
6x8	5.50 - 6.00	8.00	9.12					6.50

[a] Adapted in part from Billings and Montague. The Wartime Problem of Mica Supply: *Engineering and Mining Journal* (1944) 145, p. 94.

[b] Punch material required to yield 20 pct or more of trimmed pieces 1x1 in. or larger; price scale for larger mica based on No. 1 quality and half trim, with max bonuses of 30 pct and 40 pct for three-quarter trim and full trim, respectively.

[c] Based on full-trimmed punch and three-quarter trimmed sheet mica.

[d] Full-trimmed mica.

[e] Three-quarter trimmed mica.

[f] Based on untrimmed punch and three-quarter trimmed sheet mica.

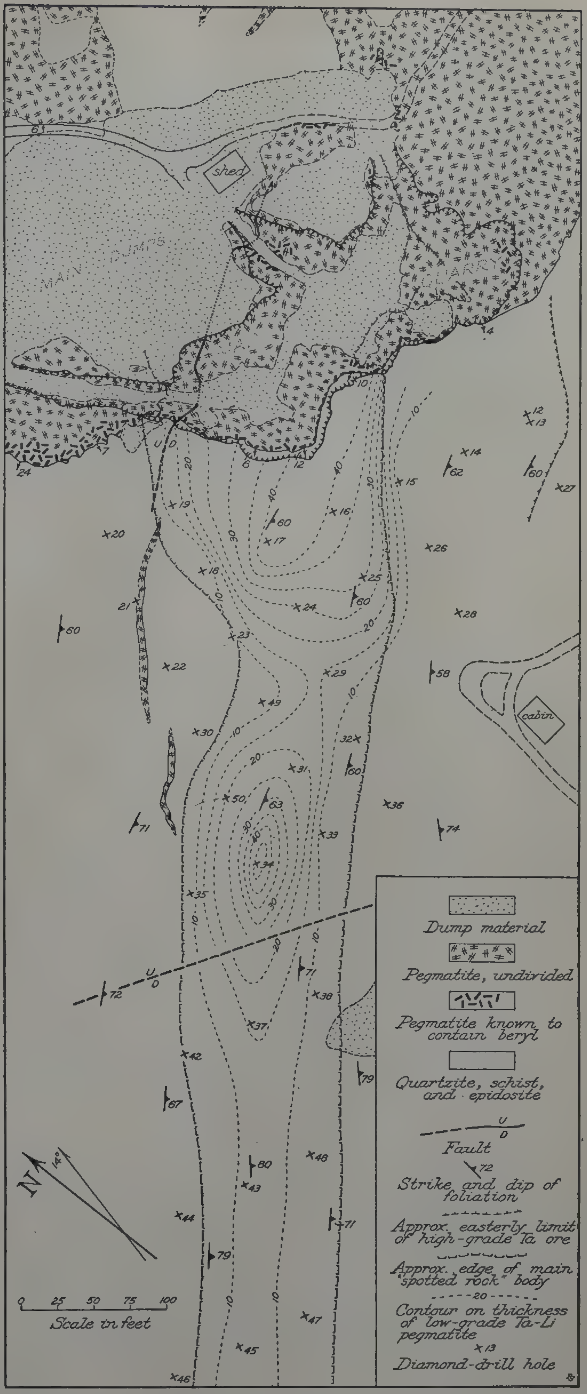
shed
MAIN DUMPS
QUARRY
cabin
Dump material
Pegmatite, undivided
Pegmatite known to contain beryl
Quartzite, schist, and epidosite
Fault
Strike and dip of foliation
Approx. easterly limit of high-grade Ta ore
Approx. edge of main "spotted rock" body
Contour on thickness of low-grade Ta-Li pegmatite
Diamond-drill hole
0 25 50 75 100
Scale in feet
N

Fig. 9—Map of a part of the Harding mine area, Taos County, N. Mex., showing distribution of U. S. Bureau of Mines diamond-drill holes and principal mass of tantalum-lithium pegmatite.

Prices for sheet mica not only fluctuate widely in response to variations in demand, but vary at any given time according to the size and quality of the material involved. The ranges for clear sheet mica are great, and during a 30-year period the price for 3 x 3 in. trimmed sheets in the southeastern United States, for example, ranged from $0.58 to $2 per lb, and that for 1½ x 2 in. trimmed sheets ranged from $0.12 to $0.60 per lb as shown in Table V. During war periods, when demands are greatly increased and problems of supply often are complex, prices characteristically reach very high levels. Thus the trend rose rapidly during the period December 1941 to December 1944, until a flat price of $8 per lb for full-trimmed sheets 2 x 2 in. and larger was in effect, see Table VI. Subsequent price scales have been considerably lower and again have been based upon size and quality of the prepared sheets.

Most of the world's sheet muscovite is obtained from India, and much smaller quantities are recovered from deposits in Brazil, Argentina, the United States, Russia, and several other countries. Domestic production rarely amounts to more than 20 pct of consumption, and during most war periods this proportion decreases distinctly. Exhaustive tests have shown that there is little intrinsic difference in electrical and other physical properties between American and Indian muscovites,[29, 30] but much of the imported material is consistently better prepared and more carefully graded.

Pegmatite Deposits

Most pegmatites can be classified as sills or dikes, depending upon whether or not they are conformable with the structure of the enclosing country rock. Variations of these forms include markedly pinching-and-swelling bodies and series of disconnected lenses and pods. In addition, troughlike, funnel-like, cigar-shaped, mushroom-shaped, and various branching forms have been recognized. Studies in all parts of the United States have shown that, despite numerous complexities of detail, most pegmatite bodies of commercial interest are rather regular in general structure. Plunging bodies are especially common, and many pegmatites that appear to be simple sills or dikes actually are shaped in three dimensions more like laths or flattened cigars, with long axes that plunge gently to moderately. Such plunging structures are extremely important as far as economic exploration of the deposits is concerned.

Pegmatite bodies that cannot be divided readily into units of contrasting composition and texture appear to constitute the great bulk of pegmatitic material in some areas. In general, however, these have received much less attention than pegmatites that are lithologically and structurally more complex. This latter group includes nearly all pegmatites that contain rare minerals, as well as most of those with minable concentrations of feldspar, mica, beryl, and other minerals. A general systematic arrangement of lithologic units in such pegmatites has long been recognized, and bands, barrels, columns, layers, lenses, pipes, pods, ribs, shoots, streaks, veins, and zones are terms commonly used by miners and referred to in geologic literature.

An essentially regular and orderly internal structure in many pegmatites was described and discussed by some earlier geologists, and even was shown on maps and sections by a few, but the attention of most investigators was directed more toward questions of mineralogy and genesis than toward structural considerations. It has remained for more recent investigators to place greater emphasis upon detailed mapping and structural interpretation of individual parts of pegmatite bodies and to demonstrate more fully the economic value of such studies in prospecting and in the planning of exploration, development, and mining.[2, 4, 5, 7–12, 31–33] In many areas it has been shown by careful studies—and in places confirmed by subsequent mining—that concentrations of economically desirable pegmatite minerals commonly occur in rock units quite distinct from adjacent barren units. Moreover, the shape and dis-

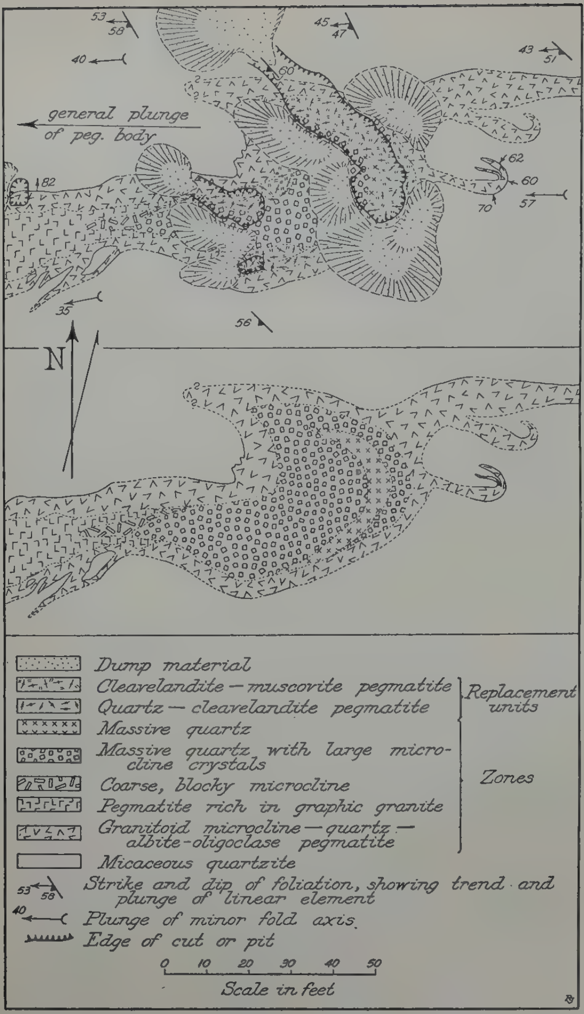
53
58
45
47
40
43
51
60
general plunge of peg. body
62
60
57
70
82
35
56
N
Dump material
Clearelandite — muscovite pegmatite
Quartz — clearelandite pegmatite
Replacement units
Massive quartz
Massive quartz with large microcline crystals
Coarse, blocky microcline
Pegmatite rich in graphic granite
Granitoid microcline — quartz — albite-oligoclase pegmatite
Zones
Micaceous quartzite
53 58 Strike and dip of foliation, showing trend and plunge of linear element
40 Plunge of minor fold axis
Edge of cut or pit
0 10 20 30 40 50
Scale in feet

Fig. 10—Geologic map of a part of the Pino Verde pegmatite, Rio Arriba County, N. Mex., with insert showing probable distribution of zones prior to albitization and development of other minerals by deuteric replacement.

tribution of many of these units reflect the general shape or structure of the host pegmatite body. The recent work thus has fully confirmed the observations and suggestions made by several earlier investigators who devoted attention to the internal structure of pegmatites.

According to a recently proposed classification,[31] the internal units of pegmatites comprise three fundamental types:

1. *Fracture fillings,* which are generally tabular bodies that fill fractures in previously consolidated pegmatite.

2. *Replacement bodies,* which are formed primarily by replacement of pre-existing pegmatite, with or without obvious structural control.

3. *Zones,* which are successive shells, complete or incomplete, that commonly reflect the shape or structure of the containing pegmatite body. Where ideally developed, they are concentric about an innermost zone, or core.

Typical pegmatite units are shown in Fig. 4. Such units range widely in size, shape, and texture. The smallest are tiny fracture-filling veinlets and the thin outermost zones of many pegmatite bodies, and the largest are masses several hundred feet long and more than 50 ft in minimum dimension. Many units are easily distinguished and sharply bounded from adjacent units, especially where they differ markedly from them in composition or texture. Contacts between others are gradational, and in some bodies of very coarse-grained pegmatite they are difficult to locate within narrow limits. Even where adjacent units are mineralogically similar, however, most boundaries can be mapped conveniently on scales of 20 or 25 ft to the inch, and independent assignments of such boundaries by more than one geologist generally agree within narrow limits.

The distribution, structure, and economic significance of many pegmatites and pegmatite units in the southwestern and southeastern United States can be most briefly explained by means of examples. The Harding pegmatites, Taos County, N. Mex., many of the pegmatites in the Petaca district, Rio Arriba County, N. Mex., and most of the commercial mica-bearing pegmatites of western North Carolina consist of two or more readily distinguishable units, and hence contain concentrations, rather than uniform disseminations, of commercially desirable minerals. The only noteworthy exceptions are in North Carolina, where much mica has been recovered from thin, rather homogeneous pegmatite lenses.

The Harding lithium-tantalum-beryllium pegmatites, in western Taos County, N. Mex., were worked during the period 1920 to 1930 for their lithium minerals, which were used in the manufacture of glass and ceramic products. Tantalum minerals were identified in several of the deposits soon after they were opened up for mining, but it was not until the early months of World War II, when Arthur Montgomery of New York City re-examined the mine openings in detail, that the abundance and commercial possibilities of these tantalum minerals were recognized. Mr. Montgomery began mining operations in December 1942, and since that time the Harding pegmatites have been the foremost domestic source of tantalum, with a total production of more than ten tons of concentrates containing an average of about 70 pct Ta_2O_5. The chief tantalum mineral is microlite, but appreciable quantities of tantalite-columbite and a little hatchettolite were included in the concentrates marketed. More than 500 lb of coarse tantalite-columbite, averaging 45 to 55 pct Ta_2O_5, was recovered from small placer deposits derived from the pegmatites.

The pegmatite of principal interest is a thick, nearly flat dike that lies within a zone of probable thrust faulting in steeply dipping metamorphic rocks of pre-Cambrian age. It dips southward, has an average thickness of 50 to 55 ft, and is known to contain beryllium, tantalum-columbium, and lithium minerals for distances of 350 ft along the strike and 650 ft down the dip. Its upper half is exceptionally well exposed along the face of a large quarry shown in Fig. 5, and its down-dip extensions were diamond drilled in 1943 and 1948 as a part of a program of thorough exploration by the U. S. Bureau of Mines.[34] This dike is strikingly layered, each layer consisting of a distinct rock type whose general position within the dike is remarkably consistent, although a given layer may show considerable variation in thickness. The layered structure has been a helpful and reliable guide in the prediction of ore-mineral distribution.

Most of the commercially recoverable beryl occurs in two 6-in. to 5-ft layers of quartz-microcline*-muscovite-albite pegmatite, one of which lies be-

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* All microcline noted in this paper contains perthitic intergrowths of sodic plagioclase.

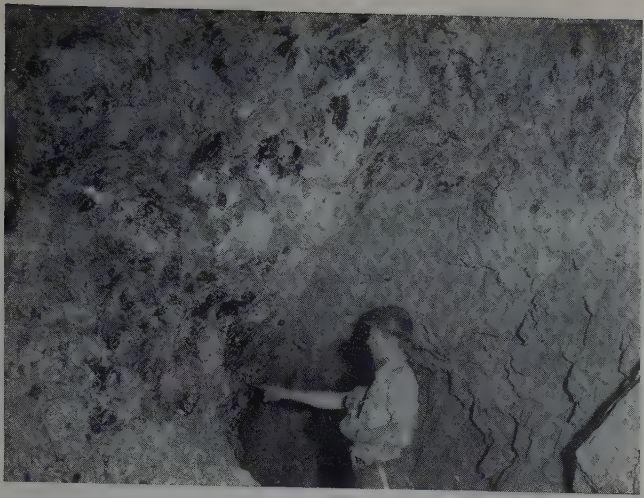


Fig. 11—Mica-bearing pegmatites, Rio Arriba County, N. Mex. Rich concentration of coarse, partly albitized mica books near pegmatite-wall rock contact, Joseph mine.

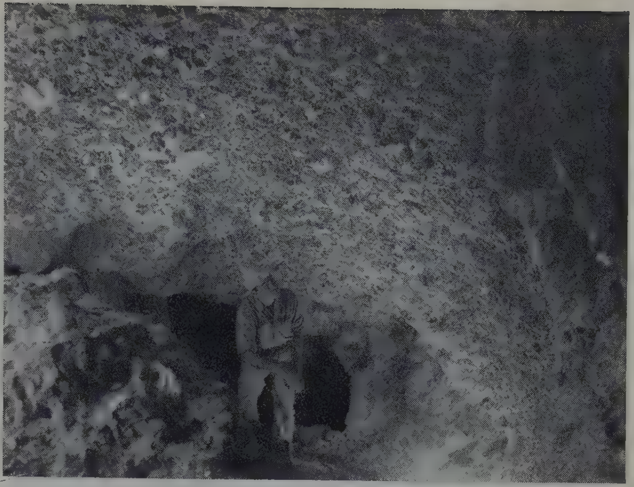


Fig. 12—Mica-bearing pegmatites, Rio Arriba County, N. Mex. Broad roll in hanging wall of Apache pegmatite. Quartz core in left foreground, remnants of mica-rich pegmatite in back of stope.

neath the hanging wall and the other above the footwall of the dike. They are best developed in its thickest parts. The beryl is white, pale yellowish green, and pinkish in color, and generally lacks crystal form. Some of the masses are extremely large, weighing 20 tons or more, and hence much of the material is readily recoverable by hand cobbing.

Immediately beneath the hanging-wall beryl-bearing layer is a somewhat thicker zone of massive quartz, which in turn grades downward into a thick zone of quartz with long, slender laths of spodumene that are spectacularly exposed on the quarry walls as shown in Fig. 5. Irregular masses of white to pinkish beryl, ½ in. to 2 ft in maximum dimension, are locally interstitial to the spodumene laths near the base of the exposed part of this unit. The beryl is associated with white to green microcline in a few places, but nowhere in the zone is either of these minerals as abundant as the quartz. The core of the dike is a coarse-grained aggregate of spodumene, microcline, and quartz, with varying quantities of albite, muscovite, lithium muscovite, lepidolite, and tantalum-columbium minerals. The pegmatite would have been symmetrically zoned at one time were not many of the original lithologic units in its lower half obscured or even obliterated by albitization, and locally through replacement by mica. The present distribution of rock types is markedly asymmetric, see Fig. 6.

The high-grade tantalum ore consists of small crystals and grains of yellow to brownish microlite and subordinate manganotantalite in a matrix of lithia mica, or locally in quartz, spodumene, muscovite, albite, or aggregates of these minerals as shown in Figs. 7 and 8. Small masses of such ore occur in a rather well-defined belt near the center of the dike. This belt appears to be 100 to 150 ft wide, and concentrations of tantalum minerals occur through a vertical range of about 15 ft. The distribution of rich ore is very irregular in detail but is surprisingly uniform within many large blocks of ground. Individual shoots have been mined by means of irregularly branching drifts, inclines, and low rooms.

Tantalite-columbite, yellowish brown to black microlite, and minor quantities of hatchettolite are disseminated throughout the central, spodumene-rich unit in the pegmatite body, see Fig. 9. This unit consists mainly of pinkish to gray, coarse but even-grained pegmatite to which the name "spotted rock" has been applied, as shown in Fig. 8. Most of the spodumene masses are nearly equidimensional, in contrast to the spodumene laths in the zone immediately above, and they average about 1 in. in diam. They are associated with microcline, albite, and quartz. Replacement of the potash feldspar and spodumene of the "spotted rock" by lepidolite has yielded large bodies of pegmatite rich in the lithia mica. The aggregates of this mica contain numerous remnants of the earlier minerals, showing all stages of replacement. Several of the lepidolite concentrations were mined out during the period 1920 to 1930.

In general the spotted rock, or low-grade tantalum-lithium-bearing pegmatite, lies immediately west of the high-grade microlite shoots. The shape of this large rock unit, as determined by observations of surface exposures and diamond-drill cores, appears to be similar to that of a loaf of French bread, with one prominent constriction and an equally prominent vertical bulge as shown in Fig. 9. Its long axis plunges 2° to 7° SW, and thus rakes slightly westward down the gentle southerly dip of the dike. The mass is about 175 ft wide and 25 to 40 ft thick, with a long, or down-plunge, dimension of at least 750 ft from the original outcrop. Its position is reflected in some places by a broad bulge in the hanging wall of the dike, and in others by a sag in the footwall, see Fig. 6.

Although much of the tantalum and some of the beryllium and lithium minerals apparently were developed by deuteric replacement of pre-existing pegmatite, their vertical distribution is broadly controlled by the layered or zonal structure of the dike. Moreover, their along-strike and down-dip distribution is in accord with discontinuities and changes in thickness of the zones. A few small fracture-controlled replacement bodies of albite and quartz with varying quantities of beryl, microlite, tantalite, hatchettolite, and bismuth minerals transect zones and zone boundaries at distinct angles, but these bodies are not abundant and are of little economic significance.

Pegmatite deposits in the Petaca district of Rio Arriba County, N. Mex., have been sources of commercial muscovite since the seventeenth century, and some of the mines may well represent the oldest systematic operations for sheet mica in this country.

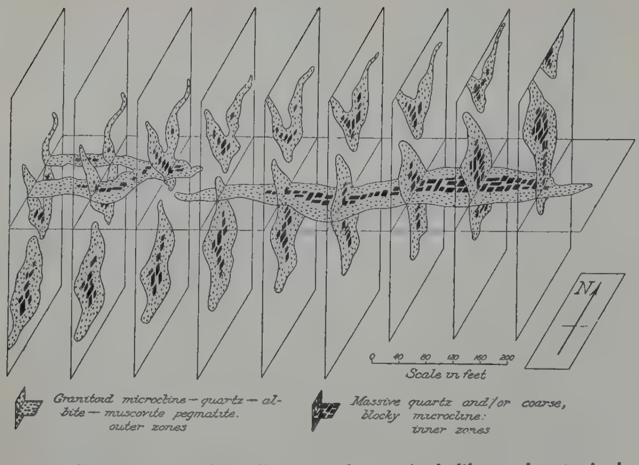

N
0 40 80 120 160 200
Scale in feet
Granitoid microcline—quartz—albite—muscovite pegmatite. outer zones
Massive quartz and/or coarse, blocky microcline: inner zones

Fig. 13—Isometric plate diagram of a typical dike and a typical troughlike pegmatite body, Petaca district, N. Mex.

These and similar deposits elsewhere in northern New Mexico have been investigated in detail during recent years by the U. S. Geological Survey, U. S. Bureau of Mines, and other organizations.[4, 12, 35-37]

The Petaca pegmatite bodies include dikes, sills, troughlike and funnel-like masses, and masses of more irregular shape, and are abundant in an elongate area of pre-Cambrian schist, quartzite, metarhyolite, and granite.[4] Most of the pegmatites are discordant, with strikingly uniform westward trends, steep dips, and moderate westward plunges. These plunges are consistently conformable with those of numerous minor structural elements in the adjacent country rock, mainly the axes of drag folds, intersections of bedding and foliation planes, and axes of "stretched" phenocrysts in the metavolcanic rocks and tectonically elongated pebbles in several conglomerate beds.

Microcline and quartz, the chief pegmatite minerals, commonly form easily distinguishable zones within individual pegmatite bodies. In general, cores of massive quartz are surrounded successively by zones of coarse, blocky microcline or of massive quartz with euhedral to subhedral microcline crystals 2 ft or more in diam, and by outer zones of fine to coarse-grained microcline-quartz pegmatite with granitoid texture as shown in Fig. 10. The border selvages of a few deposits contain albite-oligoclase. Some of the cores can be traced along their strikes into veinlike masses of quartz, and offshoots from others transect the enclosing zones and continue into the country rock as thinly tabular masses.

Superimposed upon the nearly concentric structural pattern of the zones are fracture fillings and fracture-controlled replacement bodies of quartz, cleavelandite, muscovite, and other minerals, either singly or in combinations, see Fig. 10. An association of book muscovite with albite, particularly the variety cleavelandite, is strikingly prevalent throughout the district. Little mica is present in deposits or parts of deposits that contain little albite, whereas all deposits rich in mica are also rich in the late-stage soda feldspar. The amount of muscovite in a given pegmatite unit generally is proportional to the amount of albite, but this relation does not always hold where the pegmatite consists almost entirely of albite. Such albite-rich units may contain the remnants of many mica books that appear to have been attacked and partly replaced by the plagioclase, see Figs. 11 and 12. There seems to be no correlation between the size of a given pegmatite body and its mica content. Both large and small pegmatites contain rich and extensive mica shoots, yet some of the largest dikes in the district are relatively barren of mica.

Concentrations of albite and mica are present in deposits of all general forms. They are most abundant at the keels and along the lower flanks of plunging dikes, and hence appear near the eastern ends of such bodies at their outcrops as shown in Fig. 13. They also are common at and near the keels of plunging troughlike bodies, which are U-shaped, hook-shaped, or boomerang-shaped in plan. The muscovite is thus most abundant at and near the main bends, which generally constitute the easternmost exposed parts of such pegmatites. On the other hand, the mica in plunging inverted troughlike bodies is most abundant near their western exposed parts and so is concentrated near and along their crests. In other less regular bodies, mica shoots commonly occur along bulges, indentations, "rolls," or junctions with branches, see Figs. 11 and 12.

The shapes of the mica shoots characteristically reflect the shape and structure of the enclosing zones, although in detail many shoots transect zone boundaries at distinct angles. In this latter respect the Petaca deposits are fundamentally different from most of the mica-bearing pegmatites of New England and the Southeastern States in which commercial concentrations of muscovite are largely restricted to specific zones and appear to have been developed earlier during formation of the zones.

Mica-bearing pegmatite has been mined extensively in western North Carolina for nearly 50 years, and this region has accounted for more than half the total domestic production of sheet and punch mica during that time. At least 3300 deposits are known to have been worked, and during the recent war period substantial quantities of mica were obtained from more than 1200 deposits. The Spruce Pine district, which occupies parts of Avery, Mitchell, and Yancey Counties in the Blue Ridge province, is the largest in North America, both in total production of mica and in the number of mines and prospects that lie within its limits. Since 1900 the output from at least 800 mines in this district has been marketed, and hundreds of other deposits have been prospected.

The pegmatites in North Carolina and adjacent states have been studied in considerable detail during recent years, chiefly by the U. S. Geological Survey and the U. S. Bureau of Mines.[6, 8, 9, 29, 33, 38-41] Most of the mica-bearing pegmatites occur in a metamorphic terrane that may be pre-Cambrian in age. The country rock comprises mica schist and

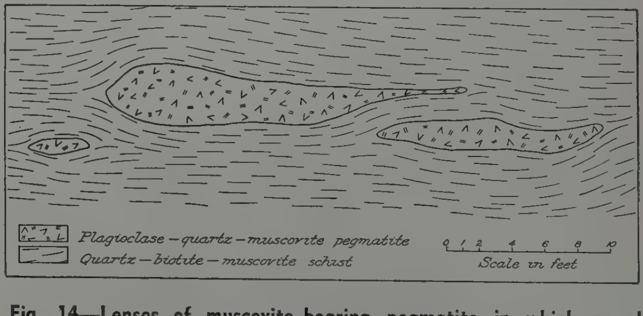

Plagioclase—quartz—muscovite pegmatite
Quartz—biotite—muscovite schist
0 1 2 4 6 8 10
Scale in feet

Fig. 14—Lenses of muscovite-bearing pegmatite in which zonal structure is not well developed.

Sketched from wall of stope in Elk mine, Avery County, N. C.

gneiss, impure quartzite, and hornblende schist and gneiss, with minor interlayered kyanite gneiss, sillimanite gneiss, graphitic schist, recrystallized limestone and dolomite, and various types of chloritic rocks. Large masses of silicic intrusive rocks, probably late Paleozoic in age, are exposed in many areas. They range in composition from quartz monzonite to quartz diorite, and commonly are surrounded and locally transected by somewhat finer-grained satellitic sills and dikes of similar composition. Most of the pegmatites that contain commercial concentrations of muscovite also are similar in composition to the large intrusive masses, and in some areas they are demonstrably related to them in genesis. Some of the pegmatites occur within, rather than adjacent to, the larger igneous masses, but they are in the minority and account for only a small proportion of the mica produced in the State.

The pegmatites are granodioritic rather than truly granitic in composition and consist of plagioclase and quartz with subordinate microcline, muscovite, biotite, and accessory minerals. Muscovite is present in some deposits as disseminated flakes, foils, and tiny books, but in others it occurs as very large books, a few of which are as much as 2 ft in diam and weigh several hundred pounds. All variations between these extremes are known, and many pegmatites contain book muscovite in a wide variety of types and forms. In general, however, the concentrations of muscovite occur within certain zones and are restricted to those zones. Thus the distribution of mica within a given deposit reflects the shape of the containing pegmatite body, although there appears to be little correlation between the size of the body and the quantity of mica within it.

Some deposits of book muscovite in the Southeastern States occur in lenses, tongues, or other pegmatite bodies that are not clearly zoned. Few such bodies are more than 6 ft thick, and many are less than 1 ft thick, see Fig 14. In general, however, the mica deposits are in well-defined units that ordinarily are quite distinct from adjacent barren units. Their distribution is clearly governed by zonal structures. Entire zones in some pegmatites are sufficiently rich in mica to be mined, but most mica deposits are confined to certain parts of zones and hence occur as shoots not unlike the shoots of ore minerals in metalliferous deposits. The position and distribution of such shoots commonly can be correlated with the overall shape of the containing zones, or with rolls, bends, constrictions, bulges, protuberances, or other irregularities in the zones, see Fig. 15.

Only a small quantity of merchantable sheet mica is associated with fracture fillings or replacement bodies, particularly in the pegmatites of greatest output. This relation is in marked contrast to that in the pegmatites of New Mexico in which nearly all the book-mica concentrations were developed in part after formation of the respective pegmatite zones that contain them. Most of the sheet muscovite produced from the Southeastern deposits is obtained from zones near the pegmatite walls. Substantial quantities also are recovered from disseminated books in poorly zoned pegmatites, especially those that are thin and lenticular, and from concentrations along the margins of quartz cores or interior zones of coarse-grained quartz and microcline.

Detailed field studies have demonstrated that the mica within a given zone is rather consistent in color, clarity, type and distribution of structural defects, and other physical properties, whereas the books from different zones within the same pegmatite commonly differ very strikingly. Green mica with numerous ridges and crenulations, or reeves, is especially abundant along the edges of quartz cores in many pegmatites, for example, whereas the mica in the earlier-formed wall zones of the same pegmatites is cinnamon brown, brown, or brownish olive, and is relatively free from structural defects. Severely reeved mica is most abundant in the pegmatites and pegmatite zones that are rich in potash feldspar, and books of the best quality generally are most numerous in oligoclase-rich zones.

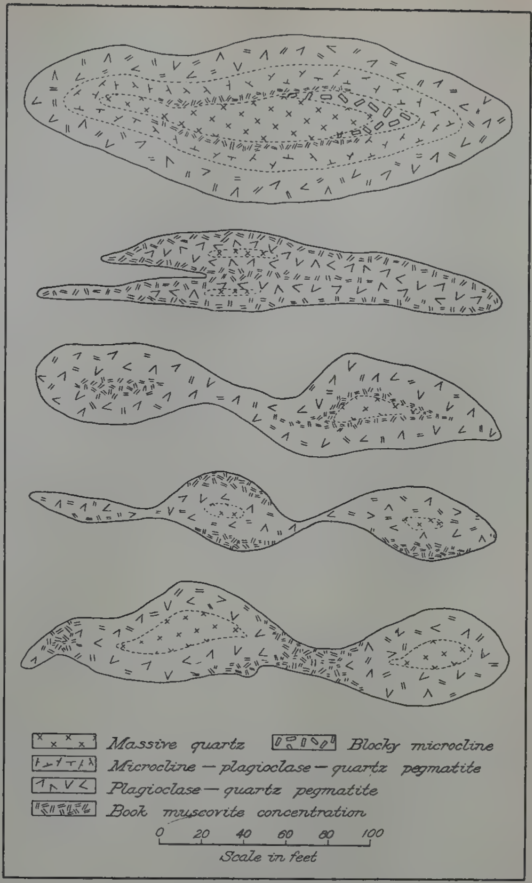

Massive quartz
Blocky microcline
Microcline — plagioclase — quartz pegmatite
Plagioclase — quartz pegmatite
Book muscovite concentration
0 20 40 60 80 100
Scale in feet

Fig. 15—Idealized plans of zoned North Carolina pegmatites showing characteristic distribution of book muscovite concentrations.

Pegmatite Mining: Operations

Most pegmatite mining is done on a small scale, with crews rarely comprising more than six men. Methods of mining and types of mine workings vary according to the size, shape, attitude, and degree of weathering of the deposits, as well as the type of labor and mining equipment available at the time.

Nearly all mines are outgrowths of prospecting operations on promising surface exposures and thus represent enlargements of trenches, shallow pits, and small, irregular cuts. As prospect openings are deepened or extended horizontally in deposits rich enough to sustain mining activities, a change to underground operations commonly is made. This is particularly common in pegmatites with moderate or gentle dips, or in other deposits where further open-cut work would require excessive timbering or the removal of much overburden. Crosscuts, drifts, stopes, and other openings are developed from adits, shafts, and inclines as the pegmatite is mined. Some

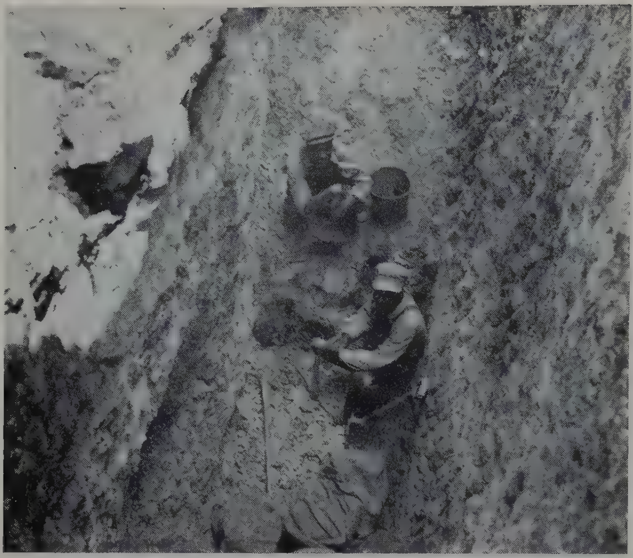


Fig. 16—Mica mining in kaolinized pegmatite.

Large mica books in weathered wall zone of White Peak No. 1 pegmatite, Powhatan County, Virginia. Quartz core is exposed on left wall of cut, country-rock schist on right wall.

of these underground workings follow definite patterns, but others are developed by highly irregular and unsystematic mining operations known as gophering or jayhawking. Such tortuous openings are especially common in the near-surface parts of weathered pegmatites.

Methods of mining are characteristically simple. The soft, decomposed pegmatite of numerous areas in the Southeastern States, for example, is easily handled with a pick and shovel. Crystals of muscovite, beryl, and other minerals that are resistant to chemical decay can be picked out of the kaolinized host rock, as shown in Figs. 16 and 17, and the waste material then can be hoisted away in buckets or small skips, dragged away in scrapers, or trammed in wheelbarrows or cars. The general simplicity of such operations has underlain the miners' preference for working in soft rock wherever possible, despite the constant danger of caving and the need for timbering. Timbering ordinarily is kept to an absolute minimum, and many of the workings collapse soon after they are opened.

Ordinary quarrying methods are employed in the mining of many unweathered pegmatites, even in some places where relatively large quantities of barren rock must be handled to permit recovery of marketable minerals. Irregular drifts, stopes, and additional underground workings are developed from surface openings in other pegmatites, and still other deposits are reached by means of shafts, inclines, or adits driven through the adjacent country rock. Drilling was done by hand in many of the older pegmatite mines, but during recent years, especially during World War II, portable compressors and mechanical drills have been widely used. Power hoists, motor-generator sets, high-capacity pumps, and other types of modern equipment are also employed in numerous current operations.

Stripping methods have found increased favor during recent years, both for removing overburden from gently dipping deposits and for eliminating dangerously steep or overhanging walls of cuts. Highly mechanized techniques, involving the use of drag-line scrapers, bulldozers, and power shovels, have proved effective at several deeply weathered deposits. One of the most spectacular operations of this type resulted in the removal of enormous quantities of country-rock schist from the gently dipping, mica-rich Big Bess pegmatite in Gaston County, North Carolina, see Fig. 17. More than 70 ft of overburden was successfully removed from parts of this pegmatite in 1944 and 1945, and very little blasting was necessary. Similar operations in some mica-bearing Brazilian pegmatites appear to have yielded satisfactory returns.[42]

Several weathered pegmatites in the Southeastern States have been mined by bulk methods, chiefly with bulldozers or drag-line scrapers. Mica and other minerals that had been left in pillars and walls of old, shallow underground mine workings were thereby recovered.

Production and Yields

The bulk of pegmatite mineral output in this country ordinarily is obtained from a relatively few mines at any given time. During the period 1942 to 1945, for example, only 83 mines in the Southeastern States qualified as very large, i.e., each of them yielded 3000 or more lb of trimmed punch and sheet muscovite. They represented only 4.5 pct of all mines and prospects in the Southeastern States from which production was obtained during that period, and yet their combined output amounted to more than 71 pct of the total from all the productive deposits as shown in Fig. 18.

In general the large and very large mica mines were operated at a profit during the period of wartime prices and other subsidies, and some of them have been satisfactory sources of commercial muscovite during other less favorable periods as well. Analyses of operating data indicate that most of these mines owe their productivity either to relatively high proportions of recoverable muscovite in the pegmatite handled, to relatively high proportions of sheet stock in the mine-run mica, or to combinations of these factors. This is in full accord with the results of similar analyses of the mica-producing districts in New England.[1] In commenting upon the yields of sheet mica from New England mines, Bannerman and Cameron[1] state that "there is no sharp division between rich and lean mines. Barring a price scale so low that no mines can operate, prices cannot be set that will eliminate the problem of marginal mines, and every revision of price during the war has brought a new group of mines into the marginal field."

The relatively large number of mica mines gives great statistical value to data bearing on their production and yields. The less abundant and widespread data that are available for beryllium, lithium, and tantalum-columbium-bearing pegmatites show similar relations. Only a few beryllium-bearing pegmatites, for example, furnish the bulk of domestic beryl production, and in general the productivity of such deposits is directly ascribable to their high yields of coarse, readily recoverable beryl per ton of rock mined. Similar relations appear to obtain in the case of other pegmatite groups, the only notable exception comprising the deposits from which relatively minor constituents are recovered as by-products.

Development of Reserves

Mining operations in pegmatite deposits have been cited repeatedly as poor financial risks, owing in large measure to uncertainties arising from the smallness and irregularity of minable concentrations of marketable minerals. Pegmatite mining traditionally has been started on promising outcrops and has been continued only until the exposed concentrations of desirable minerals were worked out or until other conditions made operations unprofitable. Thus reserves rarely are developed in advance of actual mining, and individual operations are characteristically short-lived. Moreover, the selective mining of the richest parts of mica shoots and other mineral concentrations leads to development of tortuous gopherhole workings, as well as to the leaving of much valuable material in the ground. Not only are such methods plainly wasteful, but they impose limitations on the depth to which a given shoot can be worked effectively, and they fail to uncover reserves in the form of adjacent shoots. Uncertain market conditions, limitations of available capital, and the low margin of profit commonly obtained from the sale of many pegmatite commodities militate against aggressive exploration and development of reserves. Numerous attempts have been made and a few have been conspicuously successful, but unfortunately most have antedated the accumulation of adequate structural data on the deposit being tested. In the Petaca district of New Mexico, for example, several adits and shafts were aimed at down-dip extensions of known deposits, but were developed without knowledge of the gentle to moderate plunges of those deposits. Several low-level adits that were driven to intersect productive parts of pegmatite bodies thus passed entirely beneath the keels of these bodies. Although they actually emphasize the geometric significance of the plunges involved, such disappointments nevertheless have been mentioned repeatedly in support of the thesis that pegmatite bodies are too irregular to be developed in advance of mining.

It is no accident that the exploitation of rich mica concentrations in the Petaca district downward from their outcrops has led to the development of inclines and inclined stopes that generally slope in a westward direction. It is this raking, or down-plunge, direction of mica shoots and of the pegmatite bodies themselves that plainly should receive prime consideration in the planning of future exploration and development in the district. Comparable structural features exist in other pegmatite areas, and guides for prospecting commonly can be worked out by careful analysis of each pegmatite body involved.

Outlook

The rate of discovery of new pegmatite deposits in this country has dwindled markedly during the past few decades. During the period of World War II, when prospecting and mining activities reached an all-time high, the proportion of mica and most other strategic pegmatite minerals obtained from deposits newly discovered was smaller than ever before. With the steadily increasing need for accurate appraisal of pegmatites and pegmatite groups, it has therefore become more and more necessary to focus attention on mineral concentrations not exposed at the surface and upon those only partly mined out during earlier periods of activity.

Serious problems generally confront the operator who considers reopening a pegmatite mine. The workings of many mines are inaccessible, owing to flooding, caving, or fouling with tons of backfill. For many mines, maps are not available, and reports concerning the size and distribution of workings are incomplete or inaccurate. Production and cost data rarely are complete and reliable, and often deposits must be judged solely on their general reputation among local residents. Ordinarily it is difficult to determine whether previous mining was discontinued because of intrinsic shortcomings of the deposit, poor planning or inefficiency of operations, or market conditions, or because of other factors, and it is frequently impossible to contact former operators for discussion of such matters.

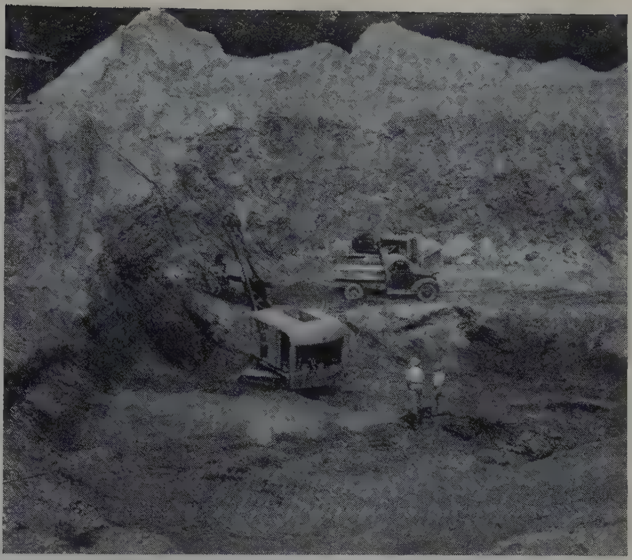


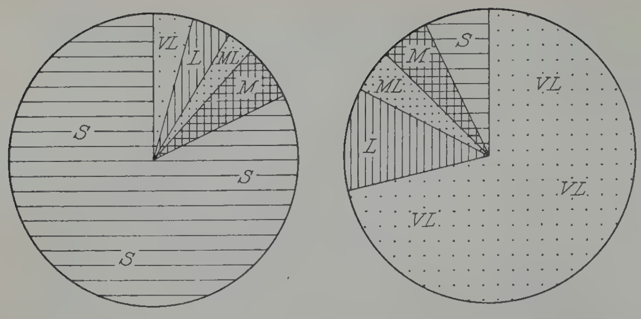
Fig. 17—Mica mining in kaolinized pegmatite.

Mechanized mining in the gently dripping Big Bess pegmatite, Gaston County, North Carolina. The pegmatite underlies the bench upon which the truck and bulldozer are standing.

Available surface exposures of pegmatites generally are little more satisfactory than mines for furnishing readily usable information. Few contain promising shoots of mica or other desirable minerals, and many are so small or discontinuous that they do not provide a clear picture of the pegmatite or its structure. Other outcrops, however, can be used to determine the internal structure of the pegmatite body and hence to determine the probable existence or distribution of mineral concentrations.

Despite the numerous difficulties involved, prediction of future possibilities for pegmatite mining is far from a hopeless task. The recent war period of high prices and subsidies led to an unprecedented expansion of pegmatite mining and prospecting in this country. Of necessity, much was done hastily and unsystematically, but opportunities for the study of pegmatites and pegmatite deposits in three dimensions were provided on a broader scale than ever before. It was possible to record and collate data on internal structure of the deposits, lithologic sequences in certain kinds of pegmatite bodies, types of book mica and other minerals, and many other features of direct or indirect economic application. More maps and detailed sections were obtained during the short period of wartime activity than had been accumulated throughout all preceding periods. The advantages of such maps and diagrams in planning and executing future operations are evident.

After they were opened during World War II, some pod and lenslike pegmatites were found to have been essentially mined out during previous periods of activity, but the mine workings in a great many other pegmatites are now known to be short of the limits of workable mineral concentrations.



Number of mines   Production of mica

Fig. 18—Classification of North Carolina mica mines according to their yields of sheet muscovite, showing the marked contrast between the relative number of small mines and their relative aggregate output.

Classification of mines, based on production during World War II:

VL — Very large, production greater than 3000 lb.
L — Large, production of 1000 to 3000 lb.
ML — Moderately large, production of 600 to 1000 lb.
M — Moderate, production of 300 to 600 lb.
S — Small, production less than 300 lb.

Large quantities of mica and other minerals were recovered from old fill, from pillars, and from thin skims of unmined pegmatite in some deposits; and recent operations in others were successful in part because of favorable market conditions and in part because of improvements in mining techniques over those previously used.

Methods of prospecting and early-stage development were much improved, especially in regions of thoroughly weathered rocks. Judicious search for pegmatite in areas of abundant float mica flakes or quartz blocks was accomplished by means of shallow trenching, deeper trenching with bulldozers or drag-line scrapers, and boring of test holes with hand augers. Such methods also were successful in many of the deeply weathered pegmatites of Brazil.[43]

The need for adequate surface exploration in advance of extensive underground development was repeatedly demonstrated, and it was further shown that the diamond drill can be an effective tool if its use is directed in close accord with known structural features of the pegmatite body being tested. Numerous blank holes bear testimony to the risks of overenthusiastic extrapolation from inadequate surface or subsurface data. Moreover, drill cores rarely provide satisfactory information on such features as the size and quality of mica books, coarseness of beryl, and general grade of the pegmatite. Nevertheless, the drill is valuable for locating extensions of known shoots or other lithologic units in pegmatites, or even for locating adjacent pegmatite bodies or shoots not elsewhere exposed and hence is useful in gathering data for rough calculations of reserves. In general, drill cores are most helpful if they are carefully interpreted in the light of structural and lithologic information from more complete exposures nearby.

Few pegmatites are completely exposed in three dimensions, so mineral shoots and incomplete zones may escape detection, either through lack of exposure or through removal by erosion, see Fig. 4. Commercial concentrations of pegmatite minerals rarely are coextensive with the pegmatite bodies that contain them, and hence they may lie undetected beneath the surface. In summarizing the structure of New England mica deposits, Bannerman and Cameron[1] state that the "pegmatites are mostly intrusive lenses, which . . . . occur . . . . at various levels with respect to the present erosion surface. Erosion has removed only the tops of some bodies, while others have been partly or almost entirely removed. If only the crest of a lens is exposed, whereas the minable mica-bearing zone is developed only along the flanks or keel, the pegmatite will be barren in surface exposures." Such relations undoubtedly are very common in most pegmatite districts,[4, 8, 9, 32] and hence hold considerable promise for future development of reserves of commercially desirable minerals.

Future operations in pegmatite districts might well be more efficient and less costly if conducted on the basis of extensive geologic data. Although there are few present criteria for recognition of workable concentrations not exposed at the surface, it commonly is possible to identify those pegmatites that offer little promise; hence they can be profitably eliminated from programs of exploration and development. The other deposits can be explored by means of trenches and test pits, and subsequently by diamond drilling and additional subsurface means if the results of the earlier work are sufficiently encouraging.

The history of mining in most pegmatite districts clearly indicates a correlation between activity and market conditions. Periods of great demand, high prices, and intensive mining are separated by longer periods of lower prices and small-scale, often sporadic, activity. The low production levels during these longer periods plainly are not caused by dwindling reserves or by suddenly increased difficulties in exposing additional shoots of commercial minerals. Thus, if market conditions appear to justify an increased production, attention might well be directed toward continuous operation of those mines known to yield the most return per unit of rock moved.

More efficient methods of mining, systematic recording of data on costs and yields for all parts of the mine,[44] the recovery of larger proportions of the mineral shoots, and the simultaneous recovery of more than one marketable product should greatly increase the chances for successful operations. More systematic and efficient mining might well be the result of a shift in attention from high-grade, pockety deposits to more extensive and continuous deposits of somewhat lower grade. The latter would lend themselves more readily to exploration in advance of mining, and efficient methods of mining and milling might lead to profitable operations. Careful mineralogic study of pegmatites and lithologic units within pegmatites is of great potential value, especially for recognition of recoverable byproducts. The possibilities for tantalum production from the Harding pegmatites, for example, might well have been recognized years ago if mineralogic studies had been carried on during the period of lepidolite mining. Studies of placer deposits, particularly those derived from pegmatites not previously worked, should yield information of possible value, so far as future mining of the heavier pegmatite minerals is concerned.

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office conferences and exchanges of information with Survey colleagues who were investigating similar deposits in other parts of the country also led to a fuller development of many general concepts and conclusions. The contributions of E. N. Cameron, J. B. Hanley, E. W. Heinrich, A. H. McNair, J. C. Olson, J. M. Parker III, and L. R. Page were particularly significant in this connection. C. S. Ross and W. T. Schaller drew generously upon their experience in previous pegmatite investigations, and provided much information and helpful criticism. The manuscript was reviewed by H. M. Bannerman, E. N. Cameron, and James Gilluly of the Geological Survey. Frances H. Jahns and Florence Wiltse gave able assistance in preparing the paper, and Joan T. Rounds drafted several of the illustrations.

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Flotation Tests on Korean Scheelite Ore

by Will Mitchell, Jr., C. L. Sollenberger, and T. G. Kirkland

The beneficiation of a scheelite ore by flotation was investigated with emphasis on determining optimum conditions for obtaining maximum grade and recovery in the rougher concentrates. Variables such as pH, time, temperature, and pulp density during conditioning were plotted against the grade and recovery of the rougher concentrates. Several gangue regulators were tried, and several methods of cleaning rougher concentrates are described.

BENEFICIATION of a Korean scheelite ore has been studied during the past year in the Basic Industries Laboratory at Allis-Chalmers Manufacturing Co. A flowsheet has been recommended including flotation, gravity separation, and a final cleaning by a magnetic separator. The investigation was continued in the field of flotation in order to establish the effect of certain variables on recovery and grade of rougher concentrates. The results of this investigation are recorded here.

Mineralogy

The scheelite ore was obtained from Korea and was mined from the Sang Dong deposit. This deposit is described as lying in gently dipping metamorphosed sedimentary beds of Cambrian age.[1] The metamorphism was probably either metasomatic replacement or recrystallization, or both, with subsequent hydrothermal deposition. Igneous bodies in the form of granite porphyry have been found within 7 km of the main shaft and are considered to be a possible source of the mineralizing solutions. The Myobong formation, the host for the Sang Dong deposit, was laid down in middle Cambrian times and consists of series of mudstone, siltstone, shale, and marl, with local stratification of mica schists, hornfels, and tactites. The scheelite is found in quartz bands in the tactites. Although there are six of these bands, only one, the "main bed" averaging 1.75 pct WO_3, has been mined to any extent.

The ore sample as received in the laboratory was the product of a cone crusher and was all −2 in. The rock appeared to be even textured, medium to fine grained, and gray green in color. No scheelite could be recognized by the unaided eye. However, under ultraviolet light of 2537 Å, the scheelite fluoresced and could be distinguished from the gangue. Under the black light, the scheelite showed no definite vein structure but was disseminated throughout the specimen.

The raw ore had a specific gravity of 2.96 and weighed about 125 lb per cu ft at −20-mesh Tyler. The −2-in. material was stage crushed through 20-mesh, and samples were cut for mineralogical, chemical, and spectrochemical analyses.

Several polished sections were cut from hand specimens. Figs. 1 and 2 show quartz to be the matrix for scheelite. It was reported by Kim[2] that the greatest scheelite concentration in the mine is found with a high concentration of quartz and a decrease in the abundance of garnet. Diameters of scheelite particles ranged from 1.2 mm down to less than 50 microns.

Spectrochemical analysis showed the ore to be high in aluminum, bismuth, calcium, iron, magnesium, manganese, silicon, tungsten, and with lesser amounts of beryllium, chromium, lithium, sodium, and titanium. Chemical assays showed the −20-

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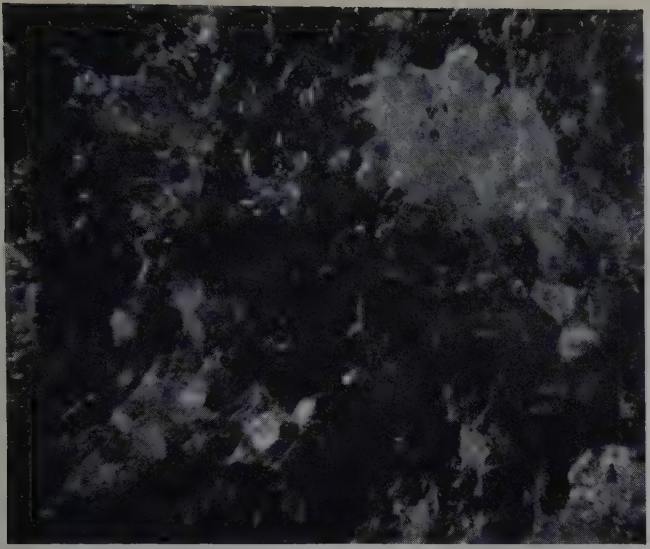


Fig. 1—Scheelite associated with quartz under reflected white light. (X35).

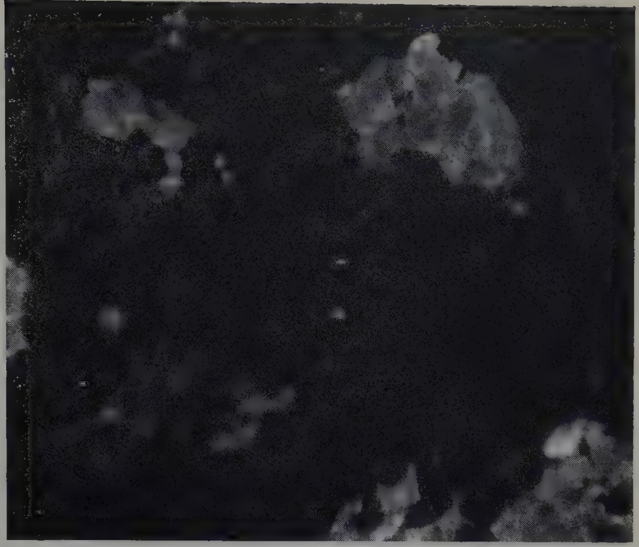


Fig. 2—Scheelite associated with quartz under reflected ultraviolet light. (X35).

mesh head to average 1.6 pct tungstate, 0.1 pct molybdate, and 0.2 pct bismuth. All tests for radioactivity were negative.

Bismuthinite and molybdenite also have been considered ore minerals, but only the flotation tests pertaining to scheelite have been included in this paper.

Table I presents a list of the significant minerals recognized in the feed, along with a rough estimate of the percentages by weight of the more abundant.

Particle counts on the individual screened fractions indicated that the scheelite was about 90 pct liberated in the —150 +200-mesh fraction. A few grains were liberated in the 20 to 28-mesh fraction. Eighty percent of the metallic minerals were liberated at 150-mesh.

The scheelite occurred as flat tabular grains, transparent and colorless. Particles fluoresced with three distinct color variations when exposed to black light. Isolation and chemical analyses of the three varieties proved that pure scheelite fluoresced blue, whereas that containing from 1 to 2 pct calcium molybdate fluoresced white, and that analyzing 4 pct or more calcium molybdate fluoresced yellow.[3] No segregation or selection of any particular variety in either the concentrates or the tails was noted when oleic acid was used as promoter.

An initial bulk sulphide float removed most of the opaque minerals; however, much of the pyrrhotite and the magnetite remained behind to be floated with the scheelite. In the sulphide float, bismuthinite appeared as elongated, sharp edged crystals, lead gray in color, and the molybdenite occurred as thin, foliated plates.

Minerals which reported in the scheelite concentrates are discussed later in the description of the floats.

Apparatus and Procedures

Flotation tests were conducted in a 500-g Fagergren test cell. The ore had been carefully stage crushed through 20-mesh and then split into approximately 500-g samples. Each sample was ground with 250 cc of distilled water in an Abbe 8x10-in. porcelain mill containing an 8-lb charge of flint pebbles. A 30-min grind was used at 58 rpm after which, the product analyzed 95.7 pct —150-mesh. The ground ore was transferred to the Pyrex glass cell and diluted to 25 pct solids with distilled water. The impeller of the test cell revolved at 1725 rpm.

Measurements of pH were made by means of a Beckman line-operated, replaceable glass electrode pH meter. Natural pH of the ore pulp in distilled water was 8.7 to 8.9.

In all tests a standard sulphide float was made to remove the sulphides, prior to scheelite flotation. The pulp was conditioned with reagent 301, 0.1 lb per ton, for 2 min; pine oil, 0.1 lb per ton, was added as a frother, and the sulphides were floated.

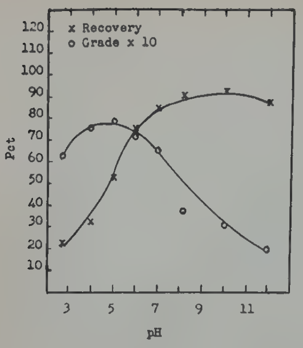
Numerous reagents have been proposed or used as collectors for scheelite. These include sodium oleate, Reagent 712, Reagent 708,[4] oleic acid, Reagent 825,[5] Orso,[6] laurylamine hydrochloride,[7] and Olate Flakes.[8] All of these reagents were investigated in this laboratory. Although slightly higher grade concentrates or higher recoveries were obtained with some of the above reagents than those obtained with oleic acid, the latter reagent was used in all the tests described in this report.

Table I. Mineralogy

Ore Minerals	Pct	Gangue Minerals	Pct	Gangue Minerals	Pct
Scheelite	1 to 2	Quartz	20 to 30	Magnetite	[illegible]
Bismuthinite	0.2	Biotite	10 to 20	Pyrrhotite	1
		Hornblende	30 to 40	Pyrite	
		Actinolite		Chalcopyrite	1
		Tremolite		Wolframite	1
		Diopside		Molybdenite	0.01
		Hedenbergite		Arsenopyrite	0.07
		Garnet	10	Galena	
		Fluorite	3	Ilmenite	
		Calcite	2		
		Apatite			
		Rutile			

Table II. Variation of Recovery and Grade with pH

pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
2.5	3.9	6.00	20.4
4.0	6.0	7.50	30.6
5.0	9.2	7.68	51.8
6.0	13.8	7.00	73.3
7.0	17.3	6.38	83.5
8.1	28.0	3.44	89.2
10.2	42.7	2.98	90.8
12.0	75.6	1.50	88.5



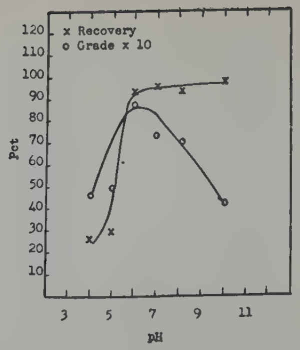
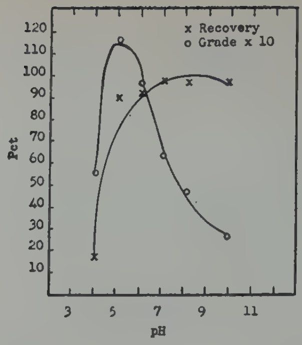


Fig. 3 (left)—Recovery and grade vs. pH. Conditioning at 26°C.

Fig. 4 (right)—Recovery and grade vs. pH. Conditioning at 58°C.



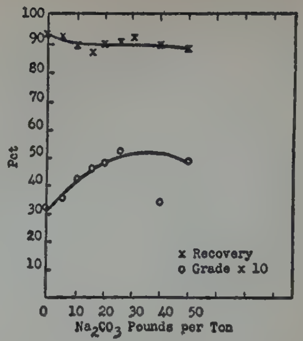


Fig. 5 (left)—Recovery and grade vs. pH. Conditioning at 80°C.

Fig. 6 (right)—Recovery and grade vs. Na_2CO_3 concentration.

To condition the pulp for several series of scheelite floats, pH or selectivity assisting reagents were added. Sufficient oleic acid was then added, (5 pct solution in wood alcohol) to obtain a concentration of 1 lb per ton. The pulp was conditioned in the test cell at 25 pct solids for 5 min. Air was then admitted and the froth collected until mineralization ceased, a period of about 5 min.

Chemical analyses of the low-grade dried products were made by the colorimetric method of Grimaldi and North.[9] High-grade concentrates were analyzed gravimetrically by a method described by Gleason.[10]

Variation of Recovery and Grade with pH

A series of floats was made in which the only variable was pH. It was varied from 3 to 12 by means of hydrochloric acid or sodium hydroxide. Conditioning was at room temperature, about 26°C. Results of these tests are listed in Table II.

These data, plotted on Fig. 3, show that maximum tungstate recovery was obtained at a pH of 10. Maximum grade was obtained at a pH of 5.

An examination of the concentrates under the petrographic microscope revealed that fluorite floated at each of the hydrogen ion concentrations investigated. At a pH of 4, gangue minerals floated were fluorite, magnetite and pyrrhotite. At pH 6, the concentrate also contained considerable biotite. At pH 10, the concentrate contained all of the foregoing gangue minerals, plus hornblende and pyroxenes.

pH Curves at Elevated Conditioning Temperatures

Since maximum recovery and maximum grade were obtained at different ends of the pH scale, it was decided to investigate the effect of elevated conditioning temperatures over the entire pH range.

These tests were made with the pulp temperatures raised to 58°C, and also to 80°C before the pH regulating agent and scheelite collector were added. The pulp was maintained at these temperatures during the 5-min conditioning period. The froth was then collected until mineralization ceased. Temperature drop during flotation was only 1 to 3 degrees. Tables III and IV show the results of these tests. Data are plotted on Figs. 4 and 5.

Table III. Variation with pH at 58°C

pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
4	7.0	4.69	25.3
5	8.7	5.00	28.4
6	14.3	8.81	91.2
7	20.9	7.31	92.1
8	16.3	7.19	91.1
10	33.7	4.00	94.5

The configuration of the recovery and grade curves obtained at the higher conditioning temperatures was similar to that obtained at room temperature; however, the maximum points increased as conditioning temperature increased. It is interesting to note that maximum grade was obtained on Fig. 3 at a pH of 5, on Fig. 4 at a pH of 6, and on Fig. 5 again at a pH of 5. The maxima of the grade and recovery curves are widely separated on Fig. 3, almost coincide on Fig. 4, and are separated again on Fig. 5. The most advantageous conditioning temperature appears to be in the range of 58°C. One can conclude that conditioning temperature, as well as pH, is critical when floating scheelite from this ore.

Variation of Addition Agents

A series of floats was made in which sodium carbonate was added in amounts varying from 0 to 50 lb per ton. Conditioning was at room temperature. The results are summarized in Table V and are plotted on Fig. 6.

Additions of sodium carbonate up to 25 to 30 lb per ton yielded proportional increases in concentrate grade but recoveries remained practically constant. Additions greater than 30 lb per ton decreased concentrate grade and recovery.

Table IV. Variation with pH at 80°C

pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
4	3.5	5.44	16.0
5	11.7	11.6	87.7
6	12.1	9.38	89.3
7	27.7	6.19	95.1
8	27.5	4.50	94.4
10	48.0	2.44	94.1

Table V. Sodium Carbonate

NA_2CO_3 Lb per Ton	pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
0	8.6	38.7	3.13	92.4
5	10.4	27.5	3.56	91.7
10	10.6	21.6	4.25	88.6
15	10.6	20.5	4.50	87.7
20	10.7	28.3	4.60	88.8
25	10.8	26.4	5.03	89.4
30	10.8	24.2	5.00	89.8
40	10.9	31.2	3.38	88.5
50	11.1	25.6	4.75	88.2

The previous tests with NaOH had shown a decrease in concentrate grade with increased pH. With Na_2CO_3, however, an increase in grade was noted as pH increased. Petrographic comparisons of concentrates obtained at pH 10 with each reagent revealed that the concentrate obtained using NaOH contained more mica and fine gangue than the concentrate obtained using Na_2CO_3. Sodium carbonate was apparently a depressant for mica and appeared to be a better dispersant of gangue minerals than sodium hydroxide.

Sodium silicate additions were varied from 0 to 9 lb per ton, while the pH was maintained at 10 with sodium hydroxide. Conditioning was at room temperature. The results obtained are listed in Table VI and are plotted on Fig. 7.

These data show that concentrate grade increased with increased concentrations of sodium silicate, whereas recovery decreased. Petrographic examinations revealed that 5 lb per ton sodium silicate depressed the quartz and the green silicate minerals, and concentrations of 7 to 9 lb per ton also depressed much of the biotite.

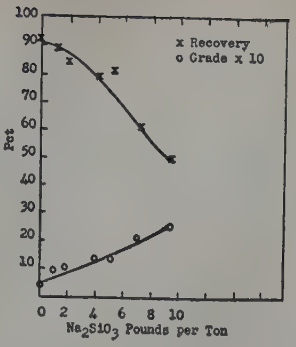
Three successive floats from a pulp containing 8 lb per ton sodium silicate were made adding 1 lb per ton oleic acid for each. It was found that 48 pct of the scheelite was recovered in the first float at a 24 pct grade, 30 pct in the second at a 14 pct grade, and 7 pct in the third at a 3.8 pct grade. This made a total recovery of 85 pct at an average grade of 14 pct WO_3. Such a scheme has the advantage of producing a relatively high-grade concentrate immediately but would probably not be desirable since it requires three stages of flotation from the bulk of the feed.

A series in which varying quantities of hydrofluoric acid were added showed that as the pH was reduced, both recovery and grade decreased until all minerals were depressed at a pH of 5.3.

A few additional rougher tests showed that a slightly increased recovery of WO_3 could be obtained if a frother having high wetting properties, such as Pentasol 124, was used instead of pine oil. A slightly increased rougher concentrate grade at essentially the same recovery could be obtained by spraying the froth with about 0.4 lb per ton Aerosol OT in the form of a 1 pct water solution.

Variation of Pulp Density During Conditioning

The effect of pulp density during conditioning was investigated in the following manner. After the sulphides were floated by the previously described method, the pulp was placed in a 2-liter beaker and decanted in successive tests to 35, 50, and 64 pct solids. To each pulp was added 1.9 lb per ton Na_2SiO_3, 1.0 lb per ton Na_2CO_3, and 1.0 lb per ton oleic acid. The pH was maintained at 10. Pulps were conditioned at room temperature for 5 min in the beaker by agitating with the impeller of the Fagergren test cell. Pentasol 124, 0.2 lb per ton, was added as a frother. The material was transferred to the test cell, diluted to 25 pct solids and floated. The results obtained are compared on Table VII and on Fig. 8.



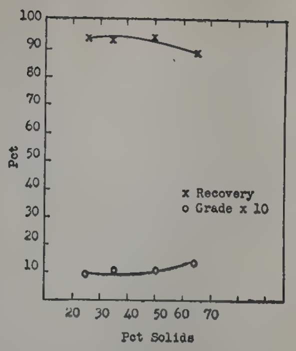


Fig. 7 (left)—Recovery and grade vs. Na_2SiO_3 concentration at pH 10.

Fig. 8 (right)—Recovery and grade vs. pulp density during conditioning.

These data show that grade increased and recovery decreased slightly with increased pulp density during the conditioning period. These effects were most pronounced when the pulp was conditioned at 64 pct solids.

Effect of Conditioning Time

From the data of the foregoing tests, it appeared that conditioning at a pulp density of 50 pct solids increased the concentrate grade with a minor decrease in recovery. A series of tests was made with pulp density during conditioning maintained at 50 pct solids, but with the conditioning time varied from 5 to 40 min. The same reagent combination was used as in the pulp density series, and conditioning was at room temperature. The results obtained are tabulated below and are plotted on Fig. 9.

A decrease in concentrate grade with increased conditioning time was recorded. Recovery remained relatively constant.

Cleaning the Rougher Concentrate

Cleaning the rougher concentrate by flotation was found to be extremely difficult because of the tenacious reagent coating on both scheelite and gangue minerals. A cleaner float without additional re-

Table VI. Sodium Silicate

Na_2SiO_3 Added Lb per Ton	pH	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
0.0	10	42.7	2.93	90.8
0.2	10	31.3	4.43	90.8
0.4	10	28.5	4.53	88.0
0.8	10	20.6	6.33	88.6
1.9	10	11.6	9.88	83.4
3.9	10	7.2	12.6	77.0
5.0	10	6.6	12.3	79.4
7.0	10	3.0	23.2	59.7
9.0	10	2.2	26.6	48.4

Table VII. Variation of Pulp Density During Conditioning

Pulp Density Pct Solids	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
25	14.2	8.94	93.3
35	16.3	9.44	92.3
50	16.3	9.69	93.0
64	11.6	12.2	89.0

Table VIII. Effect of Conditioning Time

Conditioning Time, Min	Weight Floated, Pct	WO_3 Grade, Pct	WO_3 Recovery, Pct
5	16.3	9.69	93.0
10	19.6	7.75	92.9
20	22.7	6.94	93.6
40	25.7	5.88	93.8

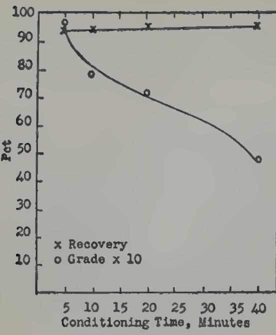


Fig. 9—Recovery and grade vs. conditioning time, conditioning at 50 pct solids.

agents increased the grade to 17.1 pct. Recovery in this concentrate dropped to 81.9 pct. However, with the addition of sodium silicate to the rougher concentrate followed by two cleaning operations, concentrate grade was increased to 27 pct. Recovery in the final concentrate without recirculation of middlings dropped to 68 pct.

A sample of rougher concentrate was roasted for 1 hr at 340°C, to remove the oleic acid coatings. By a series of cleaner floats on the roasted concentrate using sodium silicate as the regulator and about 0.6 lb per ton oleic acid in stages, a concentrate assaying 27.7 pct WO_3 was again obtained which contained 71.2 pct of the total WO_3 in the raw ore without recirculation of middlings. The concentrate in both cases contained only fluorite, scheelite and traces of magnetite and pyrrhotite.

It was found that the scheelite, after the cleaning operations, could be depressed with quebracho and the fluorite then floated with oleic acid. The fluorite float assayed only 5 pct WO_3. The nonfloat assayed 40 pct WO_3 and contained 61 pct of the WO_3 in the original ore. This was the highest concentrate grade attained by means of the flotation process.

At present the laboratory is continuing research on several reagents that appear to be much more selective to scheelite than is oleic acid.

Summary

An effort was made to show the effect of the variables usually encountered in the flotation of nonmetallic minerals on recovery and grade of rougher flotation concentrates from a complex scheelite ore, when using oleic acid as the scheelite promoter. The variables investigated were pH, conditioning time, pulp density during conditioning, conditioning temperature, and quantity of several gangue regulators.

In this investigation, the most interesting results, from an academic viewpoint, were those obtained when pH was varied at three temperature levels of conditioning. At room temperature, the peaks of the grade and recovery curves were five pH units apart. However, at 58°C, the peaks of recovery and grade percentage curves were both higher, and occurred at practically the same pH. At 80°C, still higher maximums were obtained, but the maximum points were about 2 pH units apart.

Concentrate grade equivalent to that obtained at elevated temperatures but at a slightly lower recovery was obtained with room temperature conditioning by using only 2 lb per ton sodium silicate as a gangue depressant. This reagent was the most effective gangue regulator investigated.

An increase in pulp density during conditioning caused a slight increase in grade and a slight decrease in recovery. Conditioning periods greater than 5 min caused rapid decreases in grade but had little effect upon recovery.

Rougher concentrates were cleaned to 27 pct WO_3, using additional sodium silicate as a regulator in the cleaner circuits. The cleaned concentrate was upgraded to 40 pct WO_3 by depressing the scheelite with quebracho and then floating the fluorite gangue from the scheelite.

Conclusions

From the discussion and the data recorded in this report it can be concluded that careful temperature control must be maintained during elevated temperature conditioning in a scheelite oleic acid system. Higher than room temperatures, up to a certain point, improve recovery and grade; however, beyond this point advantage is reduced. It has been proved in the tests that the pH required for optimum results varies according to the temperature of conditioning.

From an economic standpoint, elevated temperature conditioning does not appear to be practical, because equivalent grade at only slightly decreased recovery can be obtained by conditioning at room temperature and by using moderate amounts of sodium silicate as a gangue depressant.

Conditioning at high pulp densities or for periods greater than 5 min does not seem to offer any advantages. In fact, extended conditioning periods have proved deleterious to grade of concentrate.

The most satisfactory reagent combination used was found to be 1 lb per ton oleic acid, 2 lb per ton sodium silicate, 1 lb per ton sodium carbonate, 1/10 lb per ton pine oil, and 1/10 lb per ton wetting agent. A pH of 10 should be maintained if the separation is carried on at room temperature.

Acknowledgments

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High-Speed Classification and Desliming With the Liquid-Solid Cyclone

By D. A. Dahlstrom

Application of the cyclone to separation of undesirable —200-mesh clay, silt, and high-ash fractions from coal slurries was found to possess many economic and process advantages. Up to 93.3 pct removal was attained at rates of 113 to 155 gpm per sq ft of cross-sectional area.

WITHIN the past decade, certain economic, legal, and technological changes have intensified the necessity for an efficient classifier or deslimer of particles at and below the 200-mesh point. First, mechanization of mining has increased the percentage of clays, silts, refuse, and high-ash material in the run-of-mine coal. Second, many washing plants must rely on smaller percentages of fresh make-up water because of shortages, economic factors, or legal restrictions on waste water disposal. Finally, the crushing of middlings and coarse refuse plus the trends toward the finer sizes in sales has further increased the amount of fines produced per ton of mine-run coal.

A joint consequence of these circumstances has been the augmented percentage and build-up of —200-mesh material in the finer coal fractions and tipple circulating water. This results in the following disadvantages to the operator: (1) Higher ash percentages in the fine coal caused in large measure by the greater ash content of the —200-mesh material, (2) more difficult or expensive recovery of the fine coal from plant slurries, (3) lower filtration, drainage, and drying rates caused chiefly by lower permeability and the greater surface exposed per unit mass of the finer particles, (4) greater difficulty of beneficiating the fine coal because of lowered equipment efficiency and the partial "coating-out" of the —200-mesh fraction on the coarser coal particles. Obviously some cheap but effective fine-size classifier must be developed to aid in solving the above problems.

To eliminate the generally unwanted —200-mesh fraction from coal slurries, several methods are in use. Probably the most popular with coal operators is the settling cone wherein the larger particles settle downward to the apex of an inverted cone while the slimes and fines are carried upward by vertical fluid currents caused by the overflow of the bulk of the water at the outer rim. The main advantage of the unit is the low initial and operating cost since no moving parts are involved. Disadvantages are primarily low capacity per square foot or floor space and the loss of good coal in the overflow above the 100-mesh size due to localized, high-velocity gradients and nonuniform, fluid-flow patterns, especially at high loadings. Also, when high slime concentrations are present, desliming efficiency is severely injured.[1, 2]

Other methods employed are the drag and spiral classifiers and the hydroseparator.[1] These involve mechanical separations achieved by moving parts and naturally result in higher initial costs. As they are all based on the settling velocity of the particles at the mesh of separation under operating conditions, area requirements are again large per gallon of throughput if they are to classify at the 200-mesh point. Usually the selection is made to operate at a higher throughput and thus sacrifice some coarser coal in the slime overflow.

Cyclone Possibilities in Desliming

To overcome the disadvantages of present desliming methods, it is necessary to use a device that exhibits a very stable and uniform flow pattern while greatly increasing the settling velocity of

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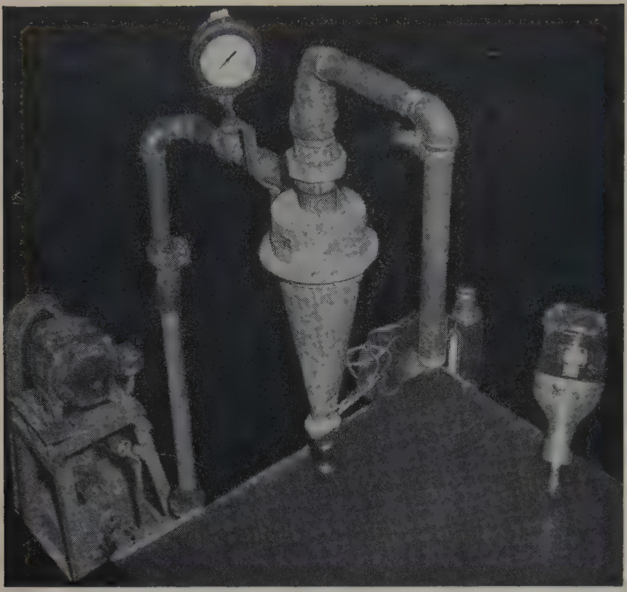

Fig. 1—Experimental desliming equipment.

the 200-mesh particle. Fortunately, the inexpensive cyclone containing no moving parts possesses both of these qualities. Basic operation and design principles have been well formulated by several authors.[3-7]

The desired symmetry and uniformity of velocity gradients within the cyclone have already been established by Criner.[8] For one specific gravity material each diameter particle will have a definite radius within the cyclone where concentration occurs. At this point the centrifugal force acting on the particle caused by the tangential velocity, radius of curvature, and particle mass is exactly balanced by the opposite frictional resistance forces due to the radial flow of the fluid past the particles toward the cyclone core. Both forces increase with a decrease in radius of curvature, but the former at a higher rate. Thus, particles coarser than the mesh of separation for the classification will tend to congregate at radii equal to, or less than, the cyclone radius, while finer particles are concentrated primarily at radii approximating the magnitude of the cyclone overflow. The result is a relatively sharp separation for particles coarser than mesh of separation when the cyclone operates at a low, underflow, volume split. Under this condition only the larger concentrating radii particles report 100 pct to the cyclone underflow while, simultaneously, the smaller radii particles are more completely carried into the cyclone overflow. Criner[8] has given a detailed discussion of the phenomenon.

Nomenclature

b = cyclone inlet diameter, in.

D = particle diameter, microns

e = cyclone overflow diameter, in.

F = cyclone energy loss, ft of fluid

G = slurry feed rate, gpm

Gpm = slurry feed rate minus underflow solids in a 50 pct slurry, gpm

K = Capacity equation proportionality constant $= [\text{gpm}/\sqrt{\text{F}}] + (\text{eb})^{0.9}$

ρ = fluid specific gravity, g per cc

ρ_s = solid specific gravity, g per cc

μ = 50 pct particle size, microns

The high centrifugal force (up to 2000 times that of gravity) that occurs within the cyclone has been pointed out by several writers.[4, 5, 7-9] This results in greatly increased settling velocities for all particles and accounts for the extremely high capacities of 200-400 gpm per sq ft of area obtainable with the cyclone. In fact, acceleration of terminal velocities is so great, the critical separation size for a cyclone operating with normal efficiencies will vary from approximately 10 to 30 microns for solids found in industrial slurries. Thus, the problem is actually to make the cyclone operate with less efficiency in order that the separation point can be raised to the 50 to 75 micron size and thereby perform a suitable desliming operation.

It has been demonstrated[4] that larger inlet and overflow nozzles, greater cyclone included angles, and smaller inlet rates all contribute toward a less efficient cyclone. Furthermore, particles, which are so fine that they experience no concentrating action, theoretically will act as a dissolved solid and thus report to underflow and overflow in the same weight ratio as the slurry water. This will be true to a lesser degree for particles slightly below the separation point. Consequently, water percent to the underflow must be held to a minimum consistent with the maintenance of the necessary vortex discharge for maximum rejection of slimes to the overflow. For coal slurries, underflow, solid content of 50 to 55 pct is probably optimum for this specification.

Some method must be found to estimate cyclone design, nozzle dimensions, and feed rates for a desliming operation. An equation for the prediction of the 50 pct point (the particle size that reports in equal weights to underflow and overflow streams), which is another name for the critical separation size or mesh of separation used in classifier terminology, had been previously obtained[4] as

$$\mu = \frac{81\,(be)^{0.68}}{(G)^{0.58}} \left[\frac{1.73}{\rho_s - \rho} \right]^{0.5} \qquad [1]$$

for a 20° cyclone with a vortex discharge. The expression is applicable to cyclone design but is so far only valid for high efficiency cyclones with 50 pct points from 8 to 30 microns. These particles appear to act according to Stokes' law under the conditions of operation, while those near the 200-mesh size will definitely deviate from this law under the increased gravitational force. Until experimental work now in progress is completed, which should develop an equation which will predict separation values accurately for desliming purposes, eq 1 must serve as a good first approximation in estimating cyclone dimensions and rates for desliming design.

The 50 pct size of eq 1 is based on an equivalent diameter particle (diameter of a sphere of same settling velocity and specific gravity as the particle in question). As coal and mineral material will be nonspherical, causing such particles to fall at a lesser rate than the same screen-size spheres, settling velocity tests should be performed if great accuracy is desired. An equivalent diameter of 75 pct to 85 pct of the actual screen diameter is a good general average. The equation was also indicated as predicting critical size values 20 pct below those determined by subsieve size distribution tests on underflow and overflow streams of industrial cyclone installations. This is due to the wide range of specific gravity solids found in such slurries, making it extremely difficult to determine size analysis accurately below the 200-mesh size. It should be emphasized that the multiplying factors do not detract from the validity of the equation. Actually, equivalent diameter is a more useful measurement than screen dimension since it accurately and simply portrays how the particle will move in a fluid slurry. The safety factor used for industrial slurries is an expediency which eliminates the necessity of breaking the —200-mesh fraction into several smaller specific gravity ranges in order to determine the 50 pct point for *each* specific gravity involved. This would be time-consuming and expensive.

Test Procedure and Equipment

To test desliming possibilities, a 9-in. cyclone was selected which had been utilized in a previous study.[4] The unit possessed interchangeable inlet, underflow, and overflow nozzles, and included angle, permitting investigation of all previously mentioned variables. It was decided to use the largest overflow and inlet nozzles (2.5 and 2.067 in., respectively) for the base comparison tests since this would give maximum aid in raising the 50 pct point predicted by eq 1. Furthermore, a better idea of capacity per unit area would be available. Also in eq 1, from experience, specific gravities of the —200-mesh fraction containing slimes and high-ash coals will run 1.7 to 2.0, so that an average value of 1.85 was selected. A fluid density of 1.0 was assumed, although this would have to be altered if high-solid-concentration feeds were used. Finally, it was decided to designate the entire —200-mesh fraction as slimes for these tests, since generally it will be desirable to eliminate totally this material. Furthermore, if the coarser particles of this fraction are desired, a higher efficiency cyclone, which is already in industrial operation, will serve this purpose. Thus, the critical size of separation will be a screen diameter of 75 microns, which must be multiplied by two safety factors of 0.8, as previously indicated. By eq 1, the flow rate necessary for the proposed cyclone can be approximated.

$$\mu = 75 \times 0.8 \times 0.8 = 48 \text{ microns}$$

$$b = 2.067 \text{ in.} \qquad \rho_s = 1.85$$

$$e = 2.5 \text{ in.} \qquad \rho = 1.0$$

and $$(G)^{0.53} = \frac{81}{48}(2.067 \times 2.5)^{0.68}\left[\frac{1.73}{1.85-1.0}\right]^{0.5}$$

from which G = 42 gpm.

Probably slightly higher feed rates can be used as any deviation from Stokes' law would tend to increase the 0.5 exponent of eq 1. Accordingly, most test work for maximum elimination of —200-mesh material was conducted at input volumes of 45 to 60 gpm.

The 9-in. cyclone was installed, as shown in Fig. 1, above a 200-gal supply tank. The cyclone was fed by a slurry pump connected to the tank through a gate valve used to throttle the flow to the desired rate. Both underflow and overflow streams returned to the tank, which was kept thoroughly mixed by a highspeed agitator. Equilibrium was easily established in less than 1 min for all runs. Test material was furnished by the Truax-Traer Coal Co. from two sources. The first was taken from the overflow of the drag tank going to the desliming tank at their Burning Star mine in Illinois, while the second originated from the slurry pond at the Fiatt, Ill., tipple at a considerable distance from the discharge pipe in order to obtain a preponderance of slimes.

Judging Desliming Efficiency

Some basis or standard must be designated in order to judge efficiencies properly. When no fresh or hydraulic water is introduced to any deslimer,

Table I. Representative Desliming Results 9 in., 20° Cyclone without Hydraulic Water

	Run 1, Burning Star Solids			Run 2, Mixed Solids		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Nozzle diam, in.	2.067	2.5	0.75	2.067	2.5	0.75
Line pressure, lb. per in.^{2}	not taken			1.56		
Volume rate, gpm	29.4	24.7	4.7	48.72	41.3	7.42
Water rate, gpm	26.85	24.2	2.65	43.75	39.6	4.15
Solid rate, tons per hr	0.97	0.23	0.74	1.83	0.68	1.15
Solid concentrate, wt pct	12.55	3.67	52.6	14.3	6.40	52.5
Ash, wt pct of solids	18.21	39.51	11.61	31.50	61.97	13.53
Solid distribution, wt pct	100.0	23.9	76.1	100.0	37.1	62.9
Ash distribution, wt pct	100.0	52.0	48.0	100.0	73.1	26.9
Water distribution, wt pct	100.0	90.1	9.9	100.0	90.5	9.5
Desliming coefficient	1.41			1.27		
Capacity ratio, K[a]				5.15		
Screen analysis, U.S. standard, wt pct						
+40-mesh	39.7	trace	52.1	34.8	trace	55.3
40x60	20.9	0.1	27.5	13.8	0.2	21.9
60x100	9.0	1.1	11.5	6.9	0.9	10.5
100x140	2.8	3.9	2.4	2.6	2.2	2.8
140x200	3.0	6.2	2.0	2.3	2.9	1.9
—200 (slimes)	24.6	88.7	4.5	39.6	93.8	7.6
Size distribution, wt pct						
+40-mesh	100.0	0.0	100.0	100.0	0.0	100.0
40x60	100.0	0.1	99.9	100.0	0.5	99.5
60x100	100.0	2.9	97.1	100.0	4.7	95.3
100x140	100.0	33.8	66.2	100.0	31.7	68.3
140x200	100.0	49.3	50.7	100.0	47.4	52.6
—200 (slimes)	100.0	86.1	13.9	100.0	87.9	12.1
Ash content, wt pct						
+40-mesh	6.29		6.29	5.78		5.78
40x60	9.40	3.71	9.42	8.70	2.67	8.71
60x100	15.28	4.54	15.53	14.68	3.78	15.31
100x140	17.00	4.84	23.61	18.02	4.39	26.49
140x200	24.92	7.29	42.10	25.92	6.24	44.24
—200	45.40	43.84	54.12	66.22	65.92	69.43

[a] K = capacity equation proportionality constant = gpm / $\sqrt{F} \div (eb)^{0.9}$, where F = cyclone energy loss, feet of fluid.

maximum elimination of a particle size to the overflow will occur when no concentration of that size takes place within the unit. The particle size then will report to the overflow and underflow stream in the same weight ratio as the water split. In this study, it is desirable to reject all the —200-mesh fraction possible, as it has been designated a slime. Thus, the first basis of desliming efficiency will be the comparison of water and —200-mesh split between overflow and underflow. Therefore, a term entitled "desliming coefficient" was used, which is merely the quotient of the percent of feed —200-mesh solids found in the underflow divided by the percent of total water reporting to the underflow. It would be desirable to make the coefficient approach zero for perfect desliming, but with no addition of hydraulic water, 1.0 will be the minimum value. While this coefficient accurately illustrates the efficiency of slime elimination under the condition of operation, it is by no means the only element to consider. Other factors, such as loss of desirable +200-mesh material to the overflow, moisture content of underflow solids, water split to underflow, feed solid concentration, cyclone capacity, and any other important operating conditions, must be given conjunctive treatment.

Test Results

Tests were performed first at normal solid loadings to the cyclone (between 12 and 15 pct solids) under the previously calculated conditions. Results are given in Table I. Run 1 was performed on the Burning Star solids, while Run 2 contained an equal amount of Fiatt material. This latter mixture was deemed more advantageous for these tests since it yielded a —200-mesh content of 40 pct by weight of total solids and thus indicates results under severe size distribution operation.

Relatively little concentration of the —200-mesh fraction in the underflow occurred as shown in Table I. Desliming coefficients of Runs 1 and 2 were 1.41 and 1.27 with the elimination of 86.1 and 87.9 pct of the slimes, respectively. Water splits were less than 10 pct to the underflow with moisture contents of both underflow solids at about the probable maximum. The higher desliming coefficient (thus the poorer desliming efficiency under operating conditions) of Run 1 was believed primarily due to the larger content of heavy quartz, calcite, and pyrite grains present near the 200-mesh size in the Burning Star solids. It would not be economically advantageous to reject such material by classification as considerably coarser coal particles would then escape to refuse. For example, let us assume a 75-micron particle of quartz (2.65 sp gr) is to be eliminated in the overflow of the classifier operated under "free settling" conditions with fluid specific gravity of 1.0. As a rough estimate of the top size of 1.4 gravity coal that would be simultaneously lost in the classifier overflow, the common classification ratio may be used.[1] Thus,

$$\frac{D_1}{D_2} = \frac{(\rho_{s2} - 1)}{(\rho_{s1} - 1)}$$

where D_1 = diameter of coal particle, microns

ρ_{s2} = 2.65 (quartz sp gr)

D_2 = diameter of quartz particle = 75 microns

ρ_{s1} = 1.4 (coal sp gr)

and $$D = 75 \frac{(2.65 - 1)}{(1.4 - 1)} = 310 \text{ microns}$$

310 microns is approximately a 45-mesh particle. Loss of substantial quantities of this coal certainly could not be tolerated. Therefore sand particles near the 200-mesh point cannot be eliminated in the overflow by any classifier without loss of some desirable coarse coal. At the same time, efficient elimination of true clays and silts possessing approximately the same specific gravity as quartz can be achieved since this material generally lies below 40 microns in size. Coarser high-ash coal in the —200-mesh fraction will be rejected to the overflow since it possesses a lower specific gravity.

Appreciable amounts of 100 x 200-mesh low-ash coal are lost to the overflow as indicated by size dis-

Table II. Representative Desliming Results 9 in., 20° Cyclone with Hydraulic Water

	Run 3, Burning Star Solids			Run 4, Burning Star Solids			Run 5, Mixed Solids			Run 6, Mixed Solids		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Nozzle diam, in.	2.067	2.5	0.75	2.067	2.5	0.75	2.067	2.5	0.75	2.067	2.5	0.75
Line pressure, lb per in.^{2}	2.0			2.0			1.88			2.0		
Volume rate, gpm	50.7	48.1	7.04	49.8	46.9	7.57	48.5	46.5	6.70	48.6	47.3	6.0
Water rate of slurry, gpm	46.8	47.5	3.97	45.6	46.0	4.3	44.2	45.1	3.8	44.8	46.1	3.44
Hydraulic water rate, gpm	4.7	(one inlet port)		4.7	(one inlet port)		4.7	(one inlet port)		4.7	(two inlet ports)	
Solid rate, tons per hr	1.37	0.29	1.08	1.42	0.29	1.13	1.67	0.69	0.98	1.58	0.68	0.90
Solid concentrate, wt pct	10.48	2.41	52.0	11.10	2.46	51.4	13.06	5.76	50.5	12.32	5.55	51.1
Ash, wt pct of solids							31.31	59.54	11.54			
Solid distribution, wt pct	100.0	21.4	78.6	100.0	20.4	79.6	100.0	41.3	58.7	100.0	43.0	57.0
Ash distribution, wt pct							100.0	78.6	21.4			
Water distribution, wt pct	100.0	92.3	7.7	100.0	91.5	8.5	100.0	92.2	7.8	100.0	93.0	7.0
Hydraulic water, pct of total water	9.1			9.4			9.6			9.5		
Desliming coefficient	1.39			1.32			0.87			0.96		
Capacity ratio, K	5.25			5.15			5.28			5.18		
Screen analysis, U.S. standard, wt pct												
+40-mesh	49.1	0.0	62.5	49.0	trace	61.5	34.4	trace	58.6	30.8	trace	54.0
40x60	17.4	trace	22.1	18.1	0.2	22.7	13.6	0.2	23.0	14.9	0.2	26.0
60x100	7.3	3.4	8.4	8.3	4.0	9.4	6.5	2.2	9.5	6.9	1.8	10.7
100x140	3.1	5.2	2.5	2.8	5.8	2.1	2.6	3.0	2.4	2.5	2.6	2.5
140x200	2.6	5.9	1.7	2.7	6.9	1.6	2.3	3.1	1.8	2.2	2.7	1.8
—200	20.5	85.5	2.8	19.1	83.1	2.7	40.6	91.5	4.7	42.7	92.7	5.0
Size distribution, wt pct												
+40-mesh	100.0	0.0	100.0	100.0	0.0	100.0	100.0	0.0	100.0	100.0	0.0	100.0
40x60	100.0	0.0	100.0	100.0	0.3	99.7	100.0	0.6	99.4	100.0	0.6	99.4
60x100	100.0	10.0	90.0	100.0	9.8	91.2	100.0	14.0	86.0	100.0	11.3	88.7
100x140	100.0	35.0	65.0	100.0	41.4	58.6	100.0	46.8	53.2	100.0	44.0	56.0
140x200	100.0	48.5	51.5	100.0	52.5	47.5	100.0	54.7	45.3	100.0	53.0	47.0
—200	100.0	89.3	10.7	100.0	88.8	11.2	100.0	93.2	6.8	100.0	93.3	6.7

tribution between underflow and overflow streams and ash analysis tests. This is due to the classification ratio phenomenon previously discussed and must occur with even the best hydraulic classifier. As a rule, the —100-mesh coal fraction represents a relatively small weight percent of the total coal, and this loss is considerably more than offset by the advantage of large slime reduction at the high capacity of the cyclone.

The ash analysis and distribution between underflow and overflow streams of Table I, Run 1 feed ash of 18.21 pct was reduced to 11.61 pct in the underflow with an overflow ash of 39.51 pct. Run 2, considerably more striking as feed ash, was 31.50 pct compared to 13.53 pct underflow and 61.97 pct overflow ash. Furthermore, almost all overflow ash is concentrated in the —200-mesh fraction with the coarser sizes being relatively pure coal. This may be expected from classification theory since the cyclone was set to discharge only high-ash slimes up to the 200-mesh point to the overflow in order to prevent serious loss of coarser coal. Sutherland[6] stated that the cyclone cleans only by removal of high-ash slimes when operating as a solid elimination or desliming unit. It is apparent that the mechanical and process advantages of elimination of the —200-mesh material is accompanied by a sizeable reduction in total ash content due entirely to the rejection of this fraction alone.

Hydraulic Water Additions

It is impossible to lower the weight distribution to the underflow of a nonconcentrated particle to less than the water split to the same stream for the cyclone under present methods of operation. However, if fresh water could be added, it might be possible partially to replace underflow slime water with new water and thus further reduce slime content. According to the theoretical and experimental work performed by Criner,[8] any fluid that reaches a certain vertical point with respect to cyclone underflow cannot reverse its path at the inner spiral but must be discharged through the underflow. The vertical distance of this point is furthermore hypothesized to be only the product of the linear distance between overflow and underflow times the fraction of total feed volume reporting to the underflow. Thus, it would seem logical to introduce fresh water slightly above this level.

To test hydraulic water possibilities, a ¼-in. pipe tangential entry with the same rotational direction as the feed nozzle was applied to the 20° cone. Location was 5¾ in. above the underflow, while vertical separation between underflow and overflow was 25½ in. Position of entry ports can be observed in Fig. 1. In the first set of runs, hydraulic water was held to less than 10 pct of the total water, while less than 8 pct of the water reported to the underflow with a solid concentration of greater than 50 pct as shown in Table II. Comparing Runs 3 and 5 with their counterparts, Runs 1 and 2, but without hydraulic water, results were found to be better in both cases. Run 5 exhibited a desliming coefficient of 0.87 (actually less than theoretical minimum, if no hydraulic water is used) compared with 1.27 for Run 2. Only 6.8 pct of the slime was retained in the underflow of Run 5, which compares favorably with 12.1 pct of Run 2. The slime percentage in Run 4 was reduced from a severe 40.6 pct in the feed down to 4.7 pct in the underflow product. Although desliming coefficients are approximately the same for Runs 1 and 3 (1.41 and 1.39, respectively), the former retained 13.9 pct of the —200-mesh solids in the underflow compared with 10.7 pct for the latter. The comparison would have been considerably better if Run 1 had been performed at a higher feed rate, since it was only 58 pct of the value for Run 3. As previously indicated, the lower feed rate is conducive to better desliming, making a strict quantitative comparison between the two runs difficult. Again, an indication of desliming possibility of the cyclone is the reduction of —200-mesh in Run 3 from 20.5 pct in the feed to 2.8 pct for the underflow. Run 4 was conducted at exactly the same conditions as Run 3 to check reproducibility. Comparison of results indicates good duplication.

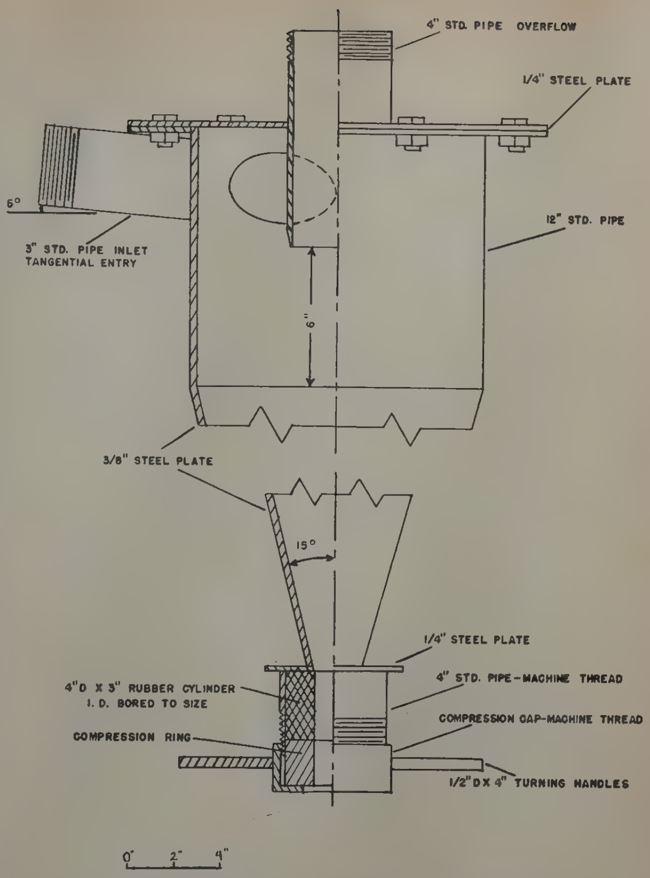
4" STD. PIPE OVERFLOW
1/4" STEEL PLATE
12" STD. PIPE
6°
3" STD. PIPE INLET TANGENTIAL ENTRY
6"
3/8" STEEL PLATE
15°
1/4" STEEL PLATE
4" STD. PIPE-MACHINE THREAD
4"D X 3" RUBBER CYLINDER I. D. BORED TO SIZE
COMPRESSION CAP-MACHINE THREAD
COMPRESSION RING
1/2"D X 4" TURNING HANDLES
0" 2" 4"

Fig. 2—Future experimental desliming cyclone with rapid adjust underflow.

As shown in Tables I and II, there is a similar effect on the coarser sizes, which reported in appreciable quantities to the overflow. Slightly smaller percentages of these sizes were recovered in the underflows of Table II, which is to be expected from classification theory. However, as the loss is limited primarily to the —100-mesh coal, it may be offset by the value of greater slime rejection.

If the addition of hydraulic water is merely a dilution of the cyclone underflow solids followed by a concentration to a higher solid content, it would be advantageous for greater slime rejection to add the same quantity of water from a larger number of ports. An attempt to prove this was made by addition of a second nozzle only 4¾ in. above the underflow. The same quantity of water was then added in Run 6, Table II, as in Run 5, but evenly divided between the two ports. Comparison of results indicates no significant advantage is present, since all factors are duplicated almost exactly. However, more thorough tests must be performed before a definite answer can be obtained. Such tests also must consider the important factors of entrance shape and location of hydraulic water ports plus input velocity and amount to be added. Probably, with maximum use of this addition, the desliming coefficient can approach very close to the perfect zero value.

Desliming Efficiency

In order to indicate qualitatively the effect of previously mentioned variables, several tests were performed, varying only one factor for each run. Runs 7, 8, and 9 of Table III were made to study the effects of solid loading and water splits. For the former variable, feed-solid concentration of the mixed solids was increased over Runs 2, 5, and 6 by approximately 100 pct without altering cyclone underflow diameter and other important design and operating factors. Run 7 was executed without addition of hydraulic water and should be compared for stability of performance with Run 2. It was indicated that heavy media and hindered settling effects were pronounced because of the high feed-solid content of 24.9 pct and an overloaded underflow discharge. The effect of the latter factor, which can be eliminated by an adjustable underflow nozzle, will be discussed under Run 9. Because of these phenomena, large percentages of 40x100-mesh material were lost in the overflow, but the desliming coefficient was reduced from 1.27 to 1.12. Only 11.9 pct of the —200-mesh fraction remained in the underflow with a solid concentration of 59.6 pct. However, it must be remembered that, while the desliming coefficient is improved by heavy solid loadings, higher percentages of water must accordingly be withdrawn in the underflow, resulting in approximately the same percentage removal of slimes as with the lower feed-solid concentrations.

Run 8 was similar to Run 7 with the exception of an addition of hydraulic water amounting to 9.4 pct of the total water entering the cyclone. Comparison of results with Run 5 indicates almost the same relationships as between Runs 7 and 2. The desliming coefficient was reduced to 0.74 with a retention of only 6.8 pct of total slimes in the underflow stream. When rejection of slimes was increased over Run 7, loss of the coarser sizes in the overflow was also slightly augmented.

To test the possibility of underflow overloading in the previous two runs and also show the disadvantage of high water splits to this stream, Run 9 was performed at similar conditions found in Run 8, but with an underflow water distribution of 36.1 pct. When the desliming coefficient remained the same for the two runs (0.74 and 0.76), 27.4 pct of the slimes reported to the underflow, having been carried there by the excess water directed to that effluent (underflow solid concentration was only 36.1 pct). This accents the necessity of retaining as high an underflow solid concentration as possible short of overloading.

Underflow overloading was present in Runs 7 and 8 since loss of +100-mesh material was considerably reduced in Run 9. This can be ascertained by comparing the lesser percentages of this fraction present in the overflow size analysis of Run 9 with the previous two runs. Apparently, the cyclone has a high degree of stability in handling varying solid loadings; but, if severe changes are to be expected, the cyclone must include an adjustable underflow.

From these results, the following conclusions may be drawn with respect to feed-solid concentration and its effect on cyclone design and installation:

(1) If solid concentration has a maximum of about 18 pct, one-stage cyclones should be used but so constructed that underflow diameter can be easily changed without shutdown in order to maintain a vortex discharge of optimum water concentration regardless of feed-solid loading. In this way, coarse coal losses can be held to a reasonable minimum. A suggested underflow design will be found in Fig. 2. It should be emphasized that underflow diameter changes will not occur often, since severe solid loading changes generally occur over long intervals. Probably, two to three simple adjustments per day will be maximum.

Table III. Effect of Feed Solid Loading and Water Split on Desliming Results, 9-in., 20° Cyclone

	Run 7, Mixed Solids			Run 8, Mixed Solids			Run 9, Mixed Solids		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Nozzle diam, in.	2.067	2.5	0.75	2.067	2.5	0.75	2.067	2.5	1.0
Line pressure, lb per in.^{2}	2.13			1.94			1.87		
Volume rate, gpm	60.3	49.5	10.8	53.9	50.0	8.0	45.4	29.6	20.5
Water rate of slurry, gpm	50.0	44.7	5.3	45.5	45.6	4.6	38.5	27.6	15.6
Hydraulic water rate, gpm	0.0			4.7	(two inlet ports)		4.7	(two inlet ports)	
Solid rate, tons per hr	4.12	2.17	1.95	3.32	1.91	1.41	2.80	0.94	1.86
Solid concentrate, wt pct	24.9	16.28	59.6	22.6	14.3	55.1	22.5	11.96	32.3
Solid distribution, wt pct	100.0	52.7	47.3	100.0	57.4	42.6	100.0	33.6	66.4
Water distribution, wt pct	100.0	89.4	10.6	100.0	90.8	9.2	100.0	63.9	36.1
Hydraulic water, pct of total water	0.0			9.4			10.9		
Desliming coefficient	1.12			0.74			0.76		
Capacity ratio, K	5.37			5.63			4.48		
Screen analysis, U.S. standard, wt pct									
+20-mesh	6.3	trace	13.4	4.8	trace	11.3	7.3	0.0	11.0
20x40	24.2	2.9	47.9	24.1	2.5	53.2	28.2	0.3	42.3
40x60	14.6	11.4	18.2	14.7	11.4	19.0	15.7	3.5	21.9
60x100	7.3	8.6	5.8	7.2	8.4	5.5	7.1	7.8	6.8
100x140	2.5	3.0	2.0	2.6	3.1	1.8	2.2	3.9	1.4
140x200	2.4	2.7	2.0	2.3	2.5	2.1	2.1	4.0	1.2
—200	42.7	71.4	10.7	44.3	72.0	7.1	37.4	80.5	15.4
Size distribution, wt pct									
+20-mesh	100.0	0.0	100.0	100.0	0.0	100.0	100.0	0.0	100.0
20x40	100.0	6.3	93.7	100.0	6.0	94.0	100.0	0.4	99.6
40x60	100.0	41.0	59.0	100.0	44.7	55.3	100.0	7.3	92.5
60x100	100.0	62.3	37.7	100.0	67.3	32.7	100.0	36.7	63.3
100x140	100.0	62.4	37.6	100.0	69.8	30.2	100.0	58.5	.41.5
140x200	100.0	60.0	40.0	100.0	61.6	38.4	100.0	62.8	37.2
—200	100.0	88.1	11.9	100.0	93.2	6.8	100.0	72.6	27.4

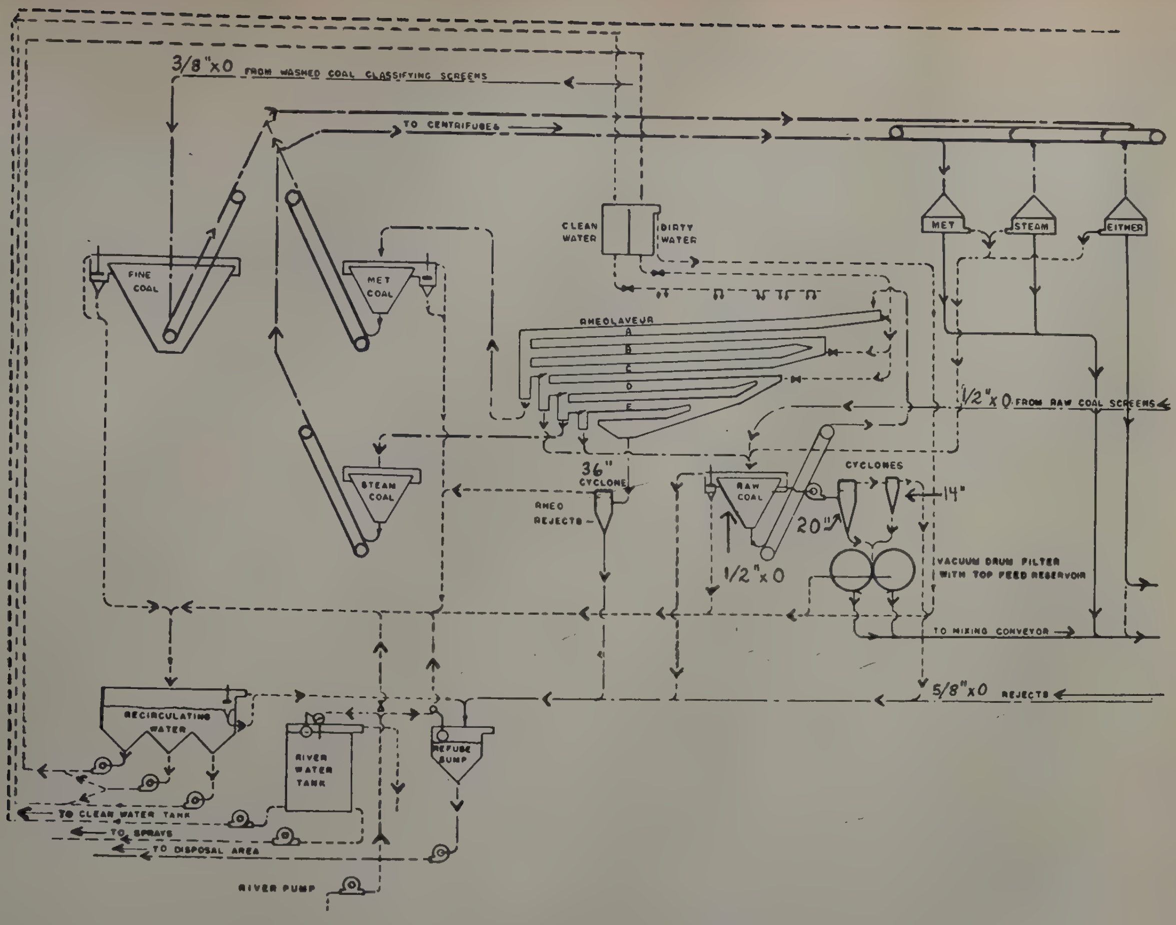

Fig. 3—Fine coal section with special cyclone desliming application.

From Sutherland

(2) When solid concentration exceeds 18 to 20 pct for appreciable time intervals, it might be most economical to install two-stage cyclones. By this means, one pump could be used to feed the first stage wherein the major portion of the coal would be withdrawn from the underflow at an optimum solid concentration. The overflow, still containing desirable material, would discharge directly to the second-stage cyclone under pressure wherein the final coal could be recovered at a similar underflow solid content. In this way, coal losses would be held to an economic minimum. Again, underflow flexibility would have to be built into both stages to permit running adjustments. This method has already been tried successfully by the Truax-Traer Coal Co. at Ceredo, W. Va.

Influence of Feed Rate and Nozzle Dimensions

Runs 10 and 11, Table IV, were performed at approximately double the feed rates of optimum Runs 2 and 5 on the same feed material. According to earlier work and eq 1, a lower 50 pct point would be predicted accompanied by a greater recovery of slimes in the underflow. Run 10 confirmed these expectations, as the desliming coefficient was 1.61 compared with only 1.27 for Run 2. As further proof of this, it will be noted that considerably higher recovery percentages of size fractions above the 200-mesh point formerly reporting to both product streams were experienced in the underflow of Run 10. Hydraulic water amounting to 7.5 pct of total feed water to the cyclone was applied to Run 11, resulting in an expected decrease of the desliming coefficient to 1.29. Slime retention, however, is still greater than that of comparable Run 5 at a lower feed rate, indicating again the higher concentrating effect of larger feed rates. In Run 11, 11.7 pct of the slimes remained in the underflow, which amounted to 7.3 pct of the total solids present in this stream, while Run 5 values were 6.8 pct and 4.7 pct, respectively. Recovery of the +200-mesh fraction in the underflow of Run 11 was quite similar to that of Run 10.

It may be concluded from the above observations that increase in slime retention of the underflow at the higher rate was only relatively slight, even though large increases in the 60x200-mesh fraction were experienced. This must be attributed to the large percentage of very fine clays and silts in the —200-mesh slimes with a correspondingly small percentage of near 200-mesh particles. If the latter material had been large, desliming efficiency would have been more severely injured in Runs 10 and 11. Thus, in order to operate at the maximum possible capacity, quantitative knowledge of the size consist of the —200-mesh fraction is a necessity. The finer the slime, the greater the capacity and the less the loss of +200-mesh coal to the overflow for any set of cyclone dimensions.

Influence of inlet and overflow nozzle dimensions was observed in the tests indicated in Table V. Conditions were maintained similar to optimum Run 5 but with the overflow nozzle reduced from 2.5 to 1.625 in. in diam. As would be predicted by eq 1,

Table IV. Effect of Feed Rate on Desliming Results, 9 in., 20° Cone

	Run 10, Mixed Solids			Run 11, Mixed Solids		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Nozzle diam, in.	2.067	2.5	1.0	2.067	2.5	1.0
Volume rate, gpm	110.6	94.0	16.6	106.7	97.5	17.0
Water rate of slurry, gpm	98.2	89.7	8.5	96.7	95.0	9.45
Hydraulic water rate, gpm	0.0			7.8	(two inlet ports)	
Solid rate, tons per hr	4.51	1.63	2.88	4.19	1.51	2.68
Solid concentrate, wt pct	15.52	6.79	57.6	14.80	6.00	52.5
Solid distribution, wt pct	100.0	36.2	63.8	100.0	36.1	63.9
Water distribution, wt pct	100.0	91.35	8.65	100.0	90.95	9.05
Hydraulic water, pct of total water	0.0			7.5		
Desliming coefficient	1.61			1.29		
Screen analysis, U.S. standard, wt pct						
+60-mesh	47.4	trace	74.3	49.2	trace	77.0
60x100	6.7	0.3	10.4	6.6	0.2	10.3
100x140	2.5	0.3	3.7	2.2	0.4	3.2
140x200	2.1	1.3	2.6	2.0	1.6	2.2
—200	41.3	98.1	9.0	40.0	97.8	7.3
Size distribution, wt pct						
+60-mesh	100.0	0.0	100.0	100.0	0.0	100.0
60x100	100.0	1.6	98.4	100.0	1.1	98.9
100x140	100.0	4.5	95.5	100.0	6.4	93.6
140x200	100.0	22.0	78.0	100.0	29.2	70.8
—200	100.0	86.1	13.9	100.0	88.3	11.7

higher underflow recoveries were experienced in all size fractions reporting to both streams. This was especially significant in the 60x140-mesh sizes. Retention of slimes in the underflow exhibited a smaller increase as the desliming coefficient was 1.18 for Run 12 as compared with 0.87 for Run 5. Furthermore, 11.1 pct of the slimes was retained in the former case, while only 6.8 pct remained in the latter. The increase would have been more severe with a coarser slime.

The inlet diameter was next changed from 2.067 to 1.25 in. in Run 13 with the overflow still at 1.625 in. At an inlet equal to that of Run 13, an even greater increase in slime retention occurred. The desliming coefficient rose significantly from 1.18 to 1.73, while practically nothing above 200-mesh reported to the overflow. The importance of the size consist of the slime is again apparent as the recovery of this fraction in the underflow actually only increased to 14.2 pct.

To demonstrate further the significance of this factor, Run 14 was performed on the Burning Star solids with smaller nozzles and higher rates than Run 1 on the same material. The desliming coefficient increased very sharply from 1.41 to 2.76 with practically nothing above 200-mesh reporting to the overflow. The run may also be compared with Run 13 as approximately the same 50 pct point would be predicted from eq 1 for both tests under the conditions of operation. Because of the coarser nature of the Burning Star slimes, a considerably higher retention of this fraction (26.2 pct) was experienced in Run 14 than with the finer slimes of Run 13 (14.2 pct). Thus, it should be re-emphasized that larger nozzles and lower feed rates are necessary with the coarser slimes.

Influence of Included Angle

Earlier studies indicated that the 50 pct point increased with an increase in cyclone included angle.[4] Accordingly, for maximum solid elimination, it is necessary to maintain included angle at the minimum permitted by economic and headroom considerations. For a classification operation, such as desliming, this factor is negated by the desire actually to discharge material in the cyclone overflow. In-

Table V. Effect of Nozzle Dimensions on Desliming Results, 9 in., 20° Cyclone

	Run 12, Mixed Solids			Run 13, Mixed Solids			Run 14, Burning Star Solids		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Nozzle diam, in.	2.067	1.625	0.75	1.25	1.625	0.75	1.75	2.0	1.0
Line pressure, lb per in.^{2}	4.88			12.0			13.5		
Volume rate, gpm	59.3	54.4	9.6	59.7	55.1	9.3	100.6	84.1	16.5
Water rate of slurry, gpm	53.7	52.9	5.5	54.3	54.1	4.85	91.3	82.6	8.7
Hydraulic water rate, gpm	4.7	(two inlet ports)		4.7	(two inlet ports)		0.0		
Solid rate, tons per hr	2.24	0.80	1.44	2.22	0.65	1.57	3.31	0.53	2.78
Solid concentrate, wt pct	14.3	5.70	51.2	14.08	4.59	56.5	12.66	2.72	56.2
Solid distribution, wt pct	100.0	35.7	64.3	100.0	29.2	70.8	100.0	16.1	83.9
Water distribution, wt pct	100.0	90.6	9.4	100.0	91.8	8.2	100.0	90.5	9.5
Hydraulic water, pct of total water	8.1			8.0			0.0		
Desliming coefficient	1.18			1.73			2.76		
Capacity ratio, K	5.65			5.65			5.17		
Screen analysis, U.S. standard, wt pct									
+60-mesh	50.6	trace	78.8				60.9	0.0	72.6
60x100	6.9	0.5	10.4				10.0	trace	11.9
100x140	2.1	1.2	2.6				3.7	0.2	4.4
+140				64.8	trace	91.5			
140x200	2.2	3.3	1.6	1.4	0.5	1.7	3.7	0.4	4.3
—200	38.2	95.0	6.6	33.8	99.5	6.8	21.7	99.4	6.8
Size distribution, wt pct									
+60-mesh	100.0	0.0	100.0				100.0	0.0	100.0
60x100	100.0	2.6	97.4				100.0	0.0	100.0
100x140	100.0	20.4	79.6				100.0	0.9	99.1
+140				100.0	0.0	100.0			
140x200	100.0	53.4	46.6	100.0	10.8	89.2	100.0	1.8	98.2
—200	100.0	88.9	11.1	100.0	85.8	14.2	100.0	73.8	26.2

stead, greater importance can be attached to maximum capacity per square foot of floor space. Theoretically, it would appear feasible to adopt larger included angles and thereby step up feed rates to obtain suitable desliming efficiency. To test this possibility, a 30° included angle cone was substituted for the 20° section used in all previous tests. Vertical distance between overflow and underflow points was 16 in. Three ¼-in. nipples were applied tangentially to the cone for entry of hydraulic water and spaced 4, 3, and 1½ in. from the underflow. Runs 15 and 16, Table VI, were then executed on the mixed material along with the indicated amounts of hydraulic water.

Results of Run 15 at slightly higher feed rates than similar Runs 5 and 6 made on the 20° cone indicated desliming efficiency to be appreciably better for the larger included angle as predicted. Desliming coefficient was 0.76 as compared with 0.87 and 0.96 for Runs 5 and 6. Furthermore the percentage of hydraulic water used in Run 15 was almost 50 pct of that for the earlier runs. Actual retention of slimes was still higher than necessary, however, because of too dilute underflow slurry. The 30° cone had a fixed underflow designed for higher operation rates and thus at this lower capacity allowed too much water to pass to the underflow.

While desliming efficiency was better, excess loss of coarser coal was evident in Run 15. This was probably due largely to (1) the high-feed solid concentration (19.95 pct) causing "hindered settling effects" as previously indicated and (2) too low an inlet velocity or capacity for the 30° cone, which has been proved to yield a higher 50 pct point at the same rate than a smaller included angle cyclone. Therefore, Run 16 was conducted with a normal feed concentration of 14.4 pct and a 20 pct increase in rate. Desliming coefficient was reduced to only 0.70, although this was probably due to the higher percentage of hydraulic water used over Run 15. Slime concentration dropped from 31.5 pct in the feed to only 3.8 pct in the product with a retention of 8.1 pct of the total slimes. Rejection of slimes would have been even better if the underflow diameter had been adjustable since product dilution was too great. Loss of coarser coal to the overflow was decreased although still appreciably larger than with the 20° cone of Runs 5 and 6. Thus, it appears that still higher capacities with good desliming efficiency can be used on the 30° cone, probably in the neighborhood of 175 to 200 pct of that with the 20° cone. The capacity advantage of the larger included angle cyclone for desliming is evident.

Industrial Correlations

Correlation of the above work with industrial data is made very difficult by the lack of such information upon actual desliming processes at the 200-mesh point. There is only one coal operation of which the author is aware being conducted on cyclones with the specific intent of rejecting all —200-mesh material possible to the overflow. Furthermore, literature data on other desliming methods on coal slurries are nonexistent.

The pilot plant operation at the Ceredo, W. Va., tipple of the Truax-Traer Coal Co. has been desliming a feed stream obtained from high levels within the raw coal cones by use of a 14-in. cyclone. A detailed discussion will be found in Sutherland's paper,[6] from which Table VII has been obtained, to illustrate cyclone performance. It will be observed that the desliming coefficients are relatively good (1.075-1.61) even though capacities are approximately three times that used on the 9-in. 20° cyclone, with comparable nozzle dimensions. The maintained desliming coefficient at the higher rates was caused by two important factors pointed out in the previous experimental work. First, Sutherland indicated that underflow solid concentration was too high, resulting in a transition to overloaded type of underflow discharge as shown in Table VII, where underflow solid concentration ranged from 55.2 pct to 60.4 pct, while a vortex discharge on this type of coal requires a 50 pct concentration. Thus, while desliming efficiency was increased through this method, it was attained with an excess loss of coarse coal to the overflow nozzle, also pointed out by Sutherland. Secondly, the slime ash content in the feed stream was generally only 30 pct, indicative of a much lighter specific gravity material than with the high-ash slimes used on the 9-in. cyclone. This also yields

Table VI. Effect of 9 in., 30° Cyclone Included Angle on Desliming Results

	Run 15, Mixed Solids			Run 16, Mixed Solids		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Nozzle diam, in.	2.067	2.5	0.5	2.067	2.5	0.5
Line pressure, lb per in.^{2}	3.13			4.88		
Volume rate, gpm	56.5	46.5	13.0	68.3	61.3	13.0
Water rate of slurry, gpm	48.8	44.2	7.6	61.9	60.0	7.85
Hydraulic water rate, gpm	3.0	(three inlet ports)		6.0	(three inlet ports)	
Solid rate, tons per hr	2.95	1.04	1.90	2.64	0.88	1.76
Solid concentrate, wt pct	19.95	8.57	49.9	14.40	5.52	47.4
Ash, wt pct of solids						
Solid distribution, wt pct	100.0	35.4	64.6	100.0	33.2	66.8
Ash distribution, wt pct						
Water distribution, wt pct	100.0	85.3	14.7	100.0	88.4	11.6
Hydraulic water, pct of total water	5.8			8.85		
Desliming coefficient	0.76			0.70		
Capacity ratio, K	4.19			4.42		
Screen analysis, U.S. standard, wt pct						
+40	41.4	trace	64.1	43.9	trace	65.7
40x60	14.5	2.0	21.3	14.2	0.7	21.0
60x100	6.6	7.0	6.3	6.6	4.9	7.4
100x140	2.2	4.3	1.0	1.7	3.3	0.9
140x200	1.8	3.1	1.2	2.1	3.8	1.2
—200	33.5	83.6	6.1	31.5	87.3	3.8
Size distribution, wt pct						
+40	100.0	0.0	100.0	100.0	0.0	100.0
40x60	100.0	4.9	95.1	100.0	1.6	98.4
60x100	100.0	37.9	62.9	100.0	24.8	75.2
100x140	100.0	70.2	29.8	100.0	64.5	35.5
140x200	100.0	59.6	40.4	100.0	61.2	38.8
—200	100.0	88.9	11.1	100.0	91.9	8.1

a lower settling velocity particle just as decreasing particle diameter. Accordingly, a further generality can be made that the lower the specific gravity of the slime, the higher the feed rate for a desliming operation.

As a corollary on the effect of solid loading on desliming efficiency, it will be observed that the desliming coefficient for the 4-ft level run was very close (1.075) to the theoretical minimum value of 1.0. Feed concentration was also 19.6 pct, the highest of the three runs and definitely caused "hindered settling" conditions. While desliming thus was increased by heavy solid loading, loss of coarser coal to the overflow was increased simultaneously, as would be expected from the experimental work.

The first information on the recently completed two-stage cyclone installation at the Ceredo plant based upon results of pilot plant operation were made available by Robert Piros, engineer for the Truax-Traer Coal Co. This process, shown in the flowsheet of Fig. 3, consists of a raw coal cone feed from the upper level to two 20-in. cyclones. The underflow discharges to final dewatering, while the overflow passes under pressure to six 14-in. cyclones. Underflow in the latter cyclones also goes to final dewatering with the overflow reporting to refuse. The first stage is operated with only a 5-lb pressure drop between inlet and overflow compared with 26 lb for the second stage. Data will be found in Table VIII and are divided into three categories, observed and assumed data and resultant calculations. Flow rates are so high for the total unit that it is presently impossible to measure flow rates, thus necessitating the assumptions and estimations. The resultant calculations are accordingly subject to large discrepancies and, therefore, can be considered only as showing trends. Desliming coefficient for the high-pressure, 14-in. cyclone was 2.46, much poorer than 1.23 for the low-pressure drop 20-in. cyclone. As pressure drop is a measure of energy available to create a centrifugal field, the higher desliming coefficient for the 14-in. cyclone seems logical. Furthermore, such a result would be predicted by eq 1.

Desliming Capacities

An empirical equation based upon experimental work and industrial correlations for any cyclone operating with a vortex underflow has been advanced in a previous article.[4]

$$\frac{\text{Gpm}}{\sqrt{F}} = K\,(be)^{0.9} \qquad [2]$$

The proportionality constant is a function only of included angle, separation between overflow and conical section, and minor design deviations. Values of 5.48 and 4.82 were found for the 20° and 30° cones, respectively, used in these tests. Assuming eq 2 as valid, *K* values were calculated for the various runs as indicated in the tables. For the 20° cyclone, the *K*'s were very consistent with the exception of Run 9. In this case, over 45 pct of the volume reported to the underflow, which undoubtedly resulted in a different flow pattern and also represents an industrially inoperable condition. Eliminating Run 9, average *K* for the 20° cone was 5.35, or only 2.4 pct below that for the higher pressure cyclone. Average value for the 30° cone on two runs was 4.31, or 10.6 pct deviation from the previous result. This small deviation is probably due to the increase in importance of wall friction when low centrifugal fields are used. If this latter force is high, practically all inlet energy is consumed in overcoming fluid friction to create a high rotational kinetic energy. It should be noted at this point that the *K* value of the 30° cone can be increased (and thus decrease energy requirements per gallon of slurry pumped) by separation of overflow and conical section. Previously it was demonstrated that by a separation of 6 in., *K* value of the 30° cone equaled or bettered the 20° cone value.[4]

Capacities per square foot of area for the desliming cyclone are extremely high when compared with the conventional classifiers. The 9-in. cyclone used in these tests yielded average rates on the mixed solids of 113 gpm of slurry per sq ft of area (exclusive of hydraulic water) for the 20° cone and 155 for the 30° cone. In the latter case, it is believed the value could have approached 220 gpm per sq ft of area without appreciable loss of desliming efficiency. Furthermore, in desliming operations, a larger percentage of the cyclone radius can be occupied by the overflow and inlet nozzles without injuring desliming efficiency, thereby increasing capacity per square foot of area. It will be observed that the proposed 12-in. desliming cyclone of Fig. 2 for future experimental work has inlet and overflow nozzles of 3 and 4-in. standard pipe, respectively. Thus, inlet and overflow nozzles consume 85 pct of the cyclone radius. For this cyclone, 200 to 235 gpm feed rate is predicted, operating on slimes of the mixed solid type used in these studies. This would offer a capacity of 250 to 300 gpm per sq ft of area for conventional methods. Actual rates for any slime will, of course, depend on the size and specific gravity of the —200-mesh fraction.

Desliming Applications

Results from the experimental work reviewed indicate that the cyclone may be applied to any desliming operation wherein up to 85 to 95 pct of the —200-mesh fraction must be removed from the coarser coal. Furthermore, this is accomplished with extremely high capacities per square foot of area, very low initial and operating expense due to the simple construction and absence of moving parts, and delivers a partially dewatered product ready for final mechanical drying.

Referring to Fig. 4 as a general practical application, any process stream which should be deslimed before final dewatering would be fed first to a deslimer cyclone. Design, nozzle dimensions, feed rate, and amount of hydraulic water would be determined by methods indicated earlier. Underflow product containing 50 to 55 pct solids could then be dewatered mechanically by use of rotary vacuum filters, continuous screen-type centrifuges, or solid bowl centrifuges. Since the underflow product is relatively fine, thus containing a high moisture content, it either would be blended back into coarser coals, if moisture restrictions permitted, or the less desirable operation of thermal drying would have to be performed.

Overflow from the deslimer, rich in unwanted fines, could follow two alternate paths. Where makeup and disposal of water is not a problem from both an economic and scarcity standpoint, the overflow can be discharged directly to waste. If a partially or fully closed water system is desired, the overflow should be processed in highly efficient cyclones to recover as much of the slimes and fines

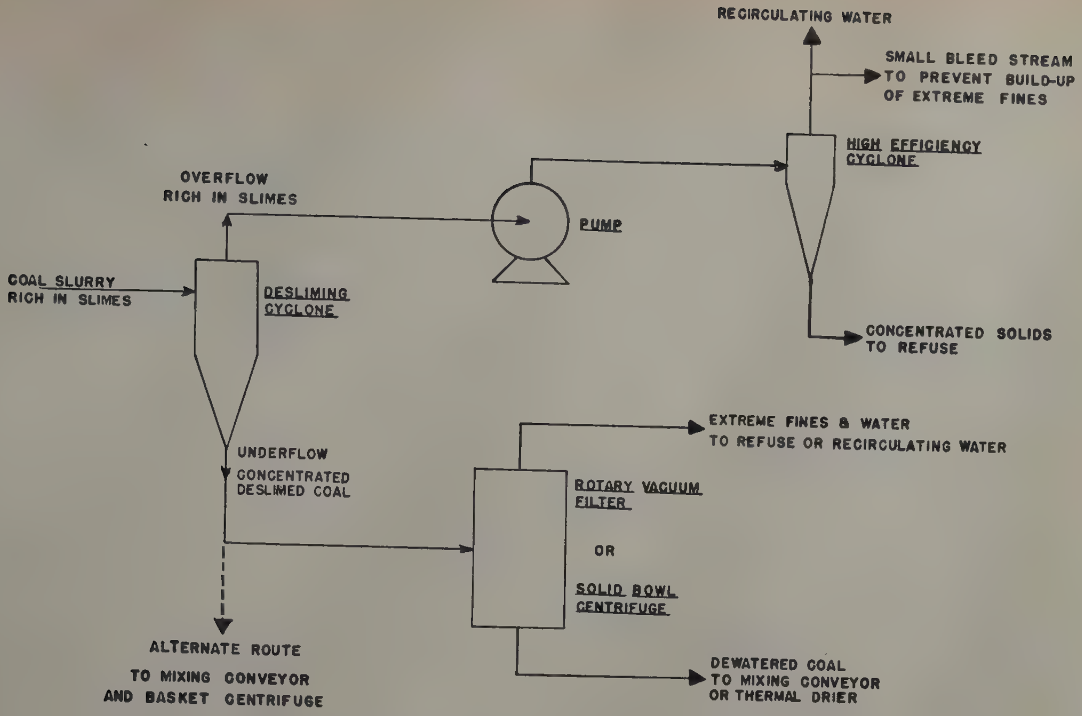
RECIRCULATING WATER
SMALL BLEED STREAM TO PREVENT BUILD-UP OF EXTREME FINES
HIGH EFFICIENCY CYCLONE
OVERFLOW RICH IN SLIMES
PUMP
COAL SLURRY RICH IN SLIMES
DESLIMING CYCLONE
CONCENTRATED SOLIDS TO REFUSE
EXTREME FINES & WATER TO REFUSE OR RECIRCULATING WATER
UNDERFLOW
CONCENTRATED DESLIMED COAL
ROTARY VACUUM FILTER
OR
SOLID BOWL CENTRIFUGE
ALTERNATE ROUTE
TO MIXING CONVEYOR AND BASKET CENTRIFUGE
DEWATERED COAL TO MIXING CONVEYOR OR THERMAL DRIER

Fig. 4—Suggested flowsheet, desliming fine coal partially closed water system.

in a concentrated form as reasonably possible, which would then be directed to waste. Methods for designing this type of cyclone were covered in earlier articles.[3, 4, 6, 7] The overflow from the secondary cyclones would be returned to the recirculating plant water with the exception of a relatively small bleed stream sent to waste to prevent buildup of extreme fines below 10 to 15 microns not adequately removed by the cyclone. This latter feature has been thoroughly discussed in the paper of Criner and Driessen, which illustrates flowsheet calculations.[8]

Future work may prove it possible to combine the cyclones of Fig. 4 into a two-stage process, eliminating the secondary pump. At present, insufficient data are available to predict resultant efficiencies for cyclones operating with an overflow back-pressure. A notable step has been made in this direction by the Truax-Traer Coal Co. in the desliming of a portion of its fine coal by drawing from high levels within the raw coal cones and feeding the slurry to desliming cyclones. In this manner, it has been possible to bypass the launder cleaning of these fines and also recover a sizeable amount of formerly wasted coal. The process has been discussed in the paper by Sutherland from which the flowsheet of Fig. 3 was taken.[6]

The possible application of the cyclone to other similar recovery programs wherein a saleable product can be recovered from a waste stream or material with relatively little expenditure must be emphasized. In light of recent developments, the author advises individual examination of all waste slurries and products to determine ash and coal concentration as a function of particle size. Many instances where slime removal from a refuse material might result in a marketable item may occur.

Optimum cyclone design for a desliming operation should include adjustable underflow diameter and maximum capacity per square foot of area. Fig. 4 pictures one design which fulfills these requirements and is now being constructed for future work at Northwestern University. The largest possible inlet and overflow diameters that can be accommodated by the cyclone radius without interference have been used. The large included angle and 6-in. separation between overflow and conical section will yield maximum capacity as previously indicated. Underflow nozzle consists of a rubber cylinder with a concentric hole bored to size. Through the cyclone design, the rubber can be compressed from three sides causing the passageway to converge at the center. Adjustment can be made speedily while the cyclone is in operation. Finally, hydraulic water ports can be added as required near the apex of the conical section. Work is now in progress to facilitate prediction of optimum entry shape, location, and fluid velocities for this particular cyclone.

Final Mechanical Dewatering Methods

As the deslimed cyclone underflow material is delivered at about 50 pct solid concentration, final means of dewatering to a reasonable moisture content are necessary. Three methods are probably most advantageous at the present writing: (1) continuous screen-type centrifuges, (2) solid bowl centrifuges, and (3) rotary vacuum filters.

The first process undoubtedly offers the cheapest initial and operating cost per ton of solids processed per hour. In operation, cyclone underflow would have to be blended with a coarser coal and fed to the centrifuges in order to prevent the loss of too much material through the screens. Furthermore, moisture content of the underflow stream is too high to permit a direct feed. Solid capacities of such units are high and thus in some instances, where unused capacity is available, new purchases may not be necessary. Several disadvantages of this method should also be indicated. The screen-type centrifuge

Table VII. Industrial 9-in., 20° Cyclone Desliming Results

Truax-Traer Coal Co., Ceredo, W. Va.

	Feed from 2-Ft Level of Raw Coal Cone			Feed from 3-Ft Level of Raw Coal Cone			Feed from 4-Ft Level of Raw Coal Cone		
	Feed	Overflow	Underflow	Feed	Overflow	Underflow	Feed	Overflow	Underflow
Nozzle diam, in.	2	2		2	2		2	2	
Pressure, lb per in.²	39			39			39		
Volume rate, gpm	191	159.5	31.5	196	166.6	29.4	195	154.9	40.1
Solid rate, tons per hr	7.67	2.72	4.95	7.81	2.64	5.47	10.21	2.66	7.55
Water rate, gpm	170.0	154.0	16.0	176.3	159.9	16.4	168.1	148.3	19.8
Solid concentrate, wt pct	15.3	6.6	55.2	15.2	6.2	57.1	19.6	6.7	60.4
Ash, wt pct of solids	10.91	18.42	6.76	11.52	19.91	7.23	9.40	17.45	6.59
—200-mesh, wt pct of solids	32.7	81.8	5.8	31.6	79.6	7.2	22.7	76.4	3.8
Solid distribution, wt pct	100	35.5	64.5	100	33.8	66.2	100	26.1	73.9
Water distribution, wt pct	100	90.5	9.5	100	90.7	9.3	100	88.2	11.8
—200-mesh distribution, wt pct	100	88.6	11.4	100	85.0	15.0	100	87.3	12.7
Desliming coefficient	1.20			1.61			1.075		

passes appreciable solids through its perforations (usually 1/16-in. round hole), which must be discarded or returned to process. The latter naturally results in a build-up of fines in the system which, coupled with the other disadvantages to follow, may lead to an aggravated condition. This type of centrifuge has an appreciable degradation factor on the coarse sizes, which causes further loss of coal and creation of slimes. Finally, the blending back of a relatively fine, wet stream with the coarser coal is known to increase severely the moisture content of the resultant product as the dewatering ability on the very fine sizes is seriously lowered. The generalization can be made that basket centrifuges become more desirable as the particle size of the deslimed coal increases.

The solid bowl centrifuge operation and results have been discussed by Lyons and Richardson.[10] The conclusions are drawn that final moisture content will be a function of percentage of —150-mesh material and water content of the centrifuge feed. A decrease in either factor will be advantageous toward a drier cake product. Dewatering capacities are very high for this difficult size, which is its prime advantage. Final moisture content is probably equal to any other method. Disadvantages seem to lie in product degradation, high initial cost, and the possible large maintenance expense due to the wearing of expensive parts.

The final method employs a rotary vacuum filter which, like the solid bowl centrifuge, is a relatively recent innovation in fine coal water elimination. Flotation concentrates have been dewatered by this process for some time, but with coarser coals, special feed methods are necessary to permit suitable formation of a filter cake. Sutherland's paper reports on the Peterson top feed reservoir rotary filter, which is gaining wider acceptance in the coal industry. Pilot-plant operations on a deslimed fine coal indicated that capacities would vary between the extremes of 6 to 39 tons of dry solids per hr per 100 sq ft of filter surface with different operating conditions. A general average of 15 to 20 tons of dry solids per hr per 100 sq ft of filter surface appears logical in this case under normal operation on a 16-mesh, top-size coal. According to Sutherland and Piros, dewatering ability will be primarily a function of feed rate and volume of air passing

Table VIII. Industrial Cyclone Desliming Results on Fine Coal Flow Circuit

Truax-Traer Coal Co., Ceredo, W. Va.

	20 In. Feed	20 In. Underflow	20 In. Overflow, 14 In. Feed	14 In. Overflow	14 In. Underflow
		Observed Data			
Pressure, lb per in.²	31.0	0.0	26.0	0.0	0.0
Solid concentrate, wt pct		48.6		5.83	54.8
Screen analysis, wt pct					
+14-mesh		3.1			
14x30		34.4			3.8
30x50		38.9		0.2	40.0
50x100		12.4		6.4	34.0
100x200		4.4		15.5	11.6
—200		6.8		77.9	10.6
Ash content, wt pct					
Total solid					
—200-mesh		8.25		15.04	7.57
Solid rate, tons per hr		30.63		18.73	32.67
Estimated vol. rate, gpm	1800			1500	21.0
		Assumptions			
Bulk, sp gr		1.17		1.02	1.18
		Resultant Calculations			
Volume rate, gpm	1800	170	1630	1500	130
Water rate, gpm	1611	102	1509	1440	69
Solid rate, tons per hr	67.5	24.2	43.3	22.3	21.0
Solid concentrate, wt pct	13.12	48.6	10.3	5.83	54.8
Water distribution, wt pct	100	6.3	93.7/100	95.4	4.6
—200-mesh distribution, wt pct	100	7.75	92.25/100	88.7	11.3
Desliming coefficient		1.23			2.46

through a unit thickness of cake. Another factor is undoubtedly size distribution, although this was found to be of less importance than predicted by previous literature on other dewatering methods. Explanation for this deviation probably can be found in the low percentage of −200-mesh in the filter feed. Literature correlations are based primarily on the content of this fraction in the feed and thus a deslimed coal could be expected to exhibit higher and more consistent rates.

Advantages of this type of filter are the complete freedom from size degradation, delivery of a product comparable in moisture content with other methods at a good capacity, high solid recovery, and low maintenance costs due to elimination of abrasive contacts. Filtrate may also be returned to the circulating water as no further slimes have been created through degradation. Disadvantages are high installation costs (about equal to the solid bowl centrifuge per ton of solid per hour) and lesser capacity per square foot of floor space than the screen-type centrifuge.

Summary and Conclusions

Experimental work has indicated that desliming or classification up to the 200-mesh point with the cyclone is not only possible but, because of the following factors, is extremely successful. Initial and operating costs of the cyclone are known to be very small because of the simplicity of design and lack of moving parts. This is especially true with the deslimer as pressures below 10 lb are used. Up to 90 to 95 pct of the slimes can be eliminated through correct cyclone construction and operation. Recovery of +100-mesh coal will be very close to 100 pct in a partially dewatered state. Cyclone operation is relatively stable even with changing feed solid concentration. If underflow flexibility is included in the design, very severe feed loading alterations can be handled. Cyclone capacities of up to 300 gal per hr per sq ft of floor space are very high when compared with the conventional values of 1 to 2 gpm per sq ft. Finally, the deslimed coal (much of which may have been wasted formerly) is of a size distribution which will exhibit reasonable, final, mechanical dewatering rates.

Although rigorous research now in progress using pure solids must be completed before exact predictions of cyclone design and feed rates can be made, eq 1 with proper safety factors affords reasonably close estimates of these values in the interim. The calculation method indicated earlier in the paper will suffice for a 20° cyclone processing a relatively coarse slime similar to the Burning Star solids. Feed rate on finer silts and clays may be increased 10 pct to 25 pct, while 30° cone capacities can be almost doubled. Energy requirements for a desliming cyclone can be predicted from eq 2 using K values approximately 95 pct of those for ordinary high pressure units. In order to gain maximum capacity, a 6-in. spacer between overflow and conical section is recommended.

The feasibility of introducing small amounts of fresh water to further reduce slime elimination has also been proved. In these tests, hydraulic water amounting to 5 to 10 pct of the total water to the cyclone resulted in a decrease of approximately 40 to 50 pct in the amount of slimes retained by the underflow product. As this innovation was originated during the tests, no precise statements as to optimum entry design and location or amounts and input velocities can be made at this time. Qualitatively, it may be stated that entry location should be at the apex of the conical section and removed from the underflow approximately by the distance obtained from multiplication of the fraction of total volume to the underflow times the vertical distance between underflow and overflow nozzles. Furthermore, inlet velocity and probably volume rate of hydraulic water must be decreased as distance from the underflow is decreased.

Prolonged feed-solid concentrations over 18 pct should be avoided for a one-stage cyclone to prevent loss of coarse coal. Methods for handling such conditions have been discussed and usually result in a two-stage flowsheet. To put the deslimer cyclone "on stream" and for running adjustments in case of severe changes in solid loading, the design should permit easy alteration of underflow diameter. One simple construction method is indicated in Fig. 2.

Possible desliming applications have been advanced together with typical flowsheets. Water recovery can be combined with this operation by use of high elimination-efficiency cyclones as indicated. Lastly, various methods of final mechanical dewatering were discussed so that entire process considerations can be accomplished.

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The Graphite of the Passau Area, Bavaria

by Russell G. Wayland

SINCE the installation at Kropfmuehl, Bavaria, of a modern flotation concentrator in 1938, the flake and fine graphite from the Passau area can now be delivered in about any normal specified carbon content of any size range up to a flake averaging about 0.7 mm. The graphite finds a wide German and export market for crucible manufacture, pencil leads, dry cells and other uses.

A controversy over the origin of the graphite deposits is being resolved in favor of syngenesis rather than epigenesis. The syngenetic theory is newly supported by the yet unpublished work of Hartmann of the Bavarian Geological Survey. Development work and exploration for graphite in the area may be changed in direction as the syngenetic theory is accepted.

Crystalline graphite is produced in the area east of Passau near the junction of Germany, Austria and Czechoslovakia, as shown in Fig. 1. This is the only graphite area of importance in Germany, and Graphitwerk Kropfmuehl AG is the only operating firm in the area. The plant and mine are located at Kropfmuehl near Hauzenberg, Kreis Wegscheid, about 10 miles east of Passau. A narrow gage railway from the mine connects with the German Railway at Schaibing Bahnhof.

The Pfaffenreuth mines date from about 1730. Until the early 20th century mining operations were carried out in a haphazard fashion. During World War I graphite mining and milling was increased, since it had to cover almost all of the crucible needs of the Central Powers. Between the wars some 11 mines were operated by two large and several small companies, but under the Nazis these were consolidated by 1938 into the Kropfmuehl enterprise. Kropfmuehl built a modern flotation mill to treat its own ores and small amounts of custom ores and tailings from the area. Since Graphitwerk Kropfmuehl AG was an I.G. Farbenindustry subsidiary, it has been under Military Government Property Control and probably will be sold to private German capital.

Geology

The country rock of the graphite area is part of the "kristallines Grundgebirge," the series of old gneissic and schistose rocks that constitutes the bed rock of most of the Bohemian basin and rims the Sudetenland. The gneissic rocks of the graphite area are considered to have been metamorphosed during the Carboniferous period. They are bordered on the north by granite stocks and penetrated by numerous smaller granite and pegmatitic intrusive rocks, as shown in Fig. 1. The gneiss is classed as a micaceous, coarse-grained cordierite gneiss by most investigators. It is much metamorphosed by the granite, particularly in the north near the larger granite bodies. Interbedded in the gneiss are the graphite seams and lenses, and also beds and lenses of crystalline limestone containing disseminated graphite in noncommercial quantities.

The gneiss, together with the included graphite and limestone seams and lenses, is cut and displaced by a number of granite sills of medium to fine grain and by a large number of diorite lamprophyre dikes and a few syenite-pegmatite dikes. The lamprophyre dikes are of various mineral compositions and textures, but many are banded and richly impregnated with pyrite; while the syenite-pegmatite dikes are coarse-grained with good crystals of titanite, pyroxene, uralite and other green amphiboles. Most investigators and the miners speak only of diorite and granite dikes cutting the graphite seams. The diorite dikes are later than the granite and some of the faulting, as is evident from Figs. 1 and 2.

Individual graphite seams and lenses may be mined for thicknesses of several feet up to several scores of feet, and for distances of several hundred feet. The aggregate thickness of a series of some 20 seams of graphite, limestone and interbedded gneiss at Kropfmuehl is stratigraphically about 450 ft. Laterally, the graphite in a seam may pinch out or grade into crystalline limestone. Graphite crystals also are found disseminated in the gneiss itself, although in unmineable concentrations. The faults that cut the graphite seams carry graphite for some feet or tens of feet away from the seams, apparently mechanically. Similarily, the graphite lenses themselves often contain mechanically introduced inclusions of wall rock, probably from flowage during folding.

Graphite crystals make up 10 to 30 pct of the fresh, mineable graphite lenses at Kropfmuehl, averaging about 20 to 25 pct after hand-sorting by the miners. In weathered lenses, the graphite concentration is said to be as high as 50 pct. The associated primary and hydrothermal minerals are dominantly feldspar and calcite, plus quartz, pyrrhotite, pyrite, biotite and occasional garnet, hornblende, sphalerite and galena. Associated secondary minerals include kaolin, nontronite, mangano-oxide-silicates (mog), adularia and chlorite. The superimposed suite of siliceous cementation minerals present consist largely of opal, chloropal, chalcedony, jasper, and hyalite. The kaolin is of special interest since it too was mined as early as 1730 and was used in the well-known Nymphenburg porcelain from 1756 on. The kaolin is derived from the gneiss and the syenite pegmatites.

The crystals of graphite vary in size within a given seam, but in seams more than a mile away from the granite on the north the average crystallinity is less coarse, lowering the commercial value. The lenses in the Kropfmuehl-Pfaffenreuth area are the most developed, and are the only ones with deep workings now accessible. Other similar crystalline graphite lenses are known from older workings at Habersdorf, Oberoetzdorf, Ficht, Diendorf, and

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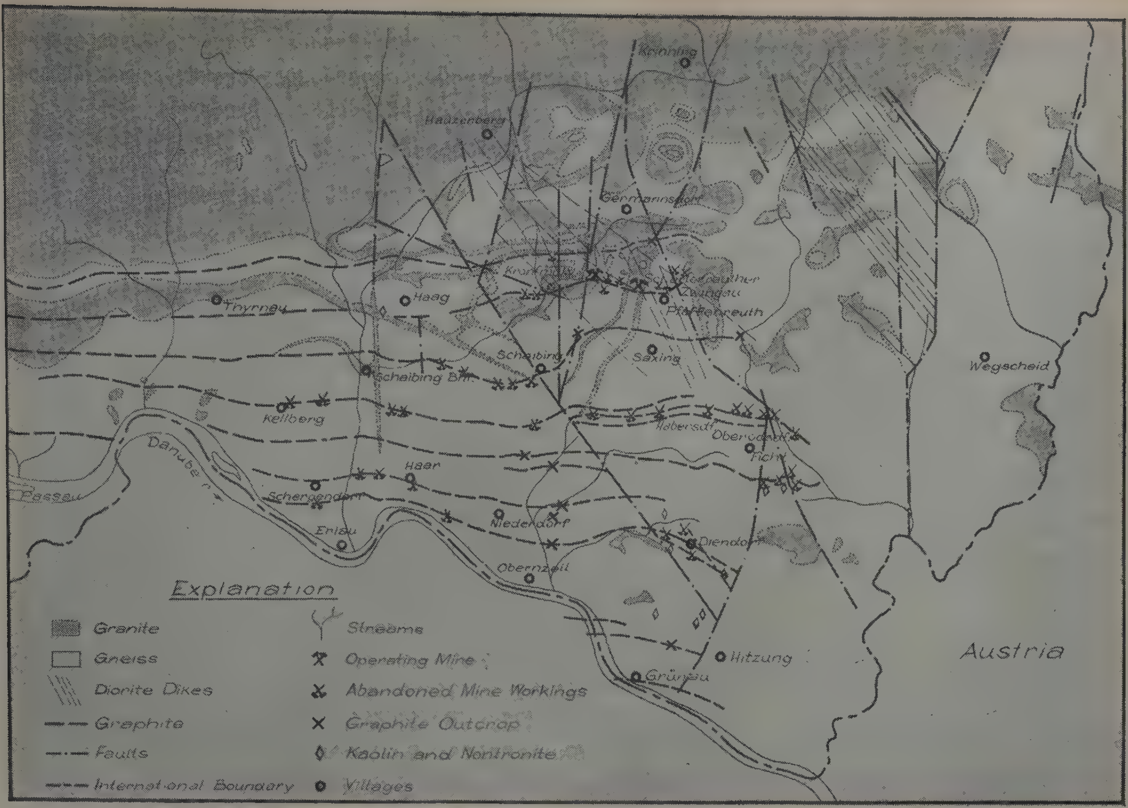

Fig. 1—The Passau graphite mining area.

Bedrock geology, after Hartmann, Bavarian Geological Survey.

Haselbach, while most of the other areas indicated on Fig. 1 await future development below the water table.

The graphite lenses and seams conform with the regional gneissic structure and follow the folding in the gneiss. The major structure in the immediate area of Kropfmuehl-Pfaffenreuth now has been shown, Fig. 2, by Hartmann to be an open syncline called the Zwingau syncline. Its axis strikes east-west, roughly parallel to the border of the granite stock to the north, and plunges gently to the west. The Zwingau saddle to the south is less well developed. Minor folding along these major structures is common. During the folding the graphite was squeezed within the original seams and also into cross fractures. The local reconcentration of graphite within a given seam has been noted to be thicker on the saddles of minor folds. Increased thickness of graphite lenses also has been correlated with the crosscutting diorite. Hartmann uses these two observations in suggesting future development work to the mine management.

Guembel,[1] the first to consider the origin of the graphite, concluded that it was syngenetic. But most German geologists since Weinschenk[2] have considered the graphite to be essentially epigenetic. Laubmann[3] summarized the development of this theory. He reasons that ascending gases and steam reacted with the gneiss, depositing graphite both by decomposition of carbon-bearing material already in the gneiss and by introduction of carbon from magmatic sources either as an unstable carbonate, a carbonyl, or a carbide. He notes that such chemical reactions to precipitate introduced carbon do not explain the apparent absence of most of the accompanying metallic ions, except in the case of the graphite crystals in the crystalline limestone where he reasons that the graphite was introduced as calcium carbide. Laubmann[3] merely mentions the possibility that the graphite might have originated in the metamorphism of richly bituminous sediments.

Hartmann is among those who consider the origin of the graphite to have been from coal or highly bituminous sediments. The shales and interbedded limestones were metamorphosed with the coal into the present rocks during the period of mountain building accompanied by the igneous activity. Subsequent hydrothermal alteration, including the kaolinization, was followed by low-temperature silicification and weathering processes.

Since the mine development in recent years has been based on the epigenetic theory, special attention has been paid to sections of the mine where it was reasoned that the graphite ascended to form replacement deposits. Acceptance of the syngenetic theory will probably redirect laterally the search for new graphite lenses.

Mining

In early mining, shafts were sunk as deep as 60 to 100 ft every 40 to 70 ft along the strike of an outcropping lens. The little underground development was by drifting and crosscutting, and workings were limited by the water table. The first modern mining was introduced shortly before World War I and extended during that war. Since the acquisition by Graphitwerk Kropfmuehl of nearly all graphite

Table I. Avg Analysis of High Quality Products, September 1948

	Crucible Flake		Best Pencil Grade
	S 40	NFL	AF
Carbon	94.5	93.0	96.2
Silica	2.22	2.98	1.63
Ferric oxide	1.35	1.41	0.93
Sulphur	0.33	0.39	0.22

Table II. Quarterly Graphite Statistics

Graphitwerk Kropfmuehl A.G.
Cleaned and Screened Graphite Concentrate.

		Production, Metric Tons			
Quarter	Year	Quarter	Pct of 1938[b]	Shipments, Metric Tons	Closing Stocks, Metric Tons
4	1945		0	N.A.	1576
1	1946	147[a]	22.9	884	839
2		829	43.1	993	675
3		1587	82.5	1214	1048
4		1222	63.5	1209	1061
1	1947	426	22.1	599	888
2		1478	76.8	1229	1139
3		1633	84.9	1739	1042
4		1393	72.4	1669	766
1	1948	1623	84.4	1776	613
2		1547	80.4	1771	389
3		1408	73.2	953	844
4		951	49.4	553	1242
1	1949	460	23.8	704	998
2		1613	83.7	967	1644
3		1269	65.9	1356	1556
4		1507	78.2	1302	1761
1	1950	1481	76.9	1471	1771
2		1181	61.3	1655	1297

[a] Production was in March only.
[b] Quarterly average for 1938 = 1925 tons.

properties, the old shafts and adits in the Kropfmuehl-Pfaffenreuth area have been connected partially with the deeper, newer workings. Three shafts are currently in use from the surface, and they have been supplemented by four blind shafts. The Kropfmuehl mine management recognizes 13 operating levels of which the sixth level, at 1310 ft elevation and 240 ft depth below the mill site, is the main haulage level at the west end of the mine. Workings extend 1.6 miles eastward from the 410-ft main shaft and have a lateral extent of as much as ¼ mile. The lowest level is the eighth at the west end at an elevation of 1120 ft, while at the Pfaffenreuth end an adit level is at 1900 ft elevation.

In the Kropfmuehl mine in 1946 there were 32 faces developed, of which 23 were being worked. The amount of graphite currently developed is enough to supply the mill for four years. The mining system is over-hand cut and fill. There are normally two miners per working face, and altogether about 200 men working underground. The blasted material is hand sorted at the face, with the waste going immediately to back-fill. Haulage on the main haulage level is by storage battery locomotive. The pumps handle 800 to 1000 gpm of water.

A new petrographic study of Landwehr[4] of the relative silicosis hazard in the rocks and ores of German mines shows that the Passau graphite ore and its dust are only about 25 pct as dangerous as the average highly siliceous rock.

Milling

According to Kausch,[5] the firm known as Gebr. Bessel (Bessel Bros.) was cleaning graphite in about 1880 by its own patented process (DRP 42). Milled graphite was mixed with 1 to 10 pct oil or fat, with or without carbon dioxide, and then boiled in water. The foam carried the graphite, while the feldspar, mica, and quartz sank. Among the reagents used were fusel oil, rubber, resin-like materials, benzine, paraffin, beeswax and ozokerite (?). To produce bubbles, they also tried about 1 pct of limestone and a little acid (DRP 39369).

By 1910, Bessel Bros. had added a process to clean the floated graphite. They screened it, mixed it with a fat (Arachis oil ?), and then worked it to a warm paste, added water and boiled it. The foam from this went to another mixer. Finally the oil was distilled off at 1100°C and the graphite again washed. The last concentrator using this process was built before World War I, and it enabled the mining and milling of fresh unweathered ores of lower grade than previously had been mined. Graphite recovery in 1908 in this mill was 734 metric tons of refined graphite or 15 pct of the crude ore mined. An additional 354 metric tons of low quality also were recovered.

Between the World Wars the concentration of Passau graphite was by a process still in use in southern Bohemia and recently described by Gruender.[6] After fine milling in the presence of normal flotation reagents and alkali, the graphite ore in a suspension about 8 pct solid was directed at 6 atm pressure onto the surface of the water in the middle of a large tank about 10 x 26 ft in size. The tank had a trough-like bottom where air bubbles entered and nongraphite minerals were drawn off. A graphite froth formed on the surface in the middle of the tank and moved to skimming points at both ends. Nongraphite minerals settled out of the froth as it moved over the more quiet parts of the tank toward the skimming points. In Bohemia the concentrate is reprocessed twice, which increases the concentration respectively from about 52 to 71 to 92 pct graphite, and the practice at Passau is understood to have been similar.

Beneficiation in the present Kropfmuehl mill (Fig. 3) is based on modern flotation cells which use both mechanical agitation and introduced air. The pulp is not introduced under pressure. The processing of the ore takes place in four distinct steps, crushing, concentrating, drying, and dry milling. In the crushing department a Krupp jaw crusher is followed by trommels, picking belt and a Symons cone crusher set to about ⅛ in. In the concentrator this material is taken from a bin and passes through a rod mill in closed circuit with a screw classifier which delivers all the ore at about 40-mesh to the 23 flotation cells. The graphite is floated from the pyrrhotite and pyrite, which constitute about 4 to 5 pct of the raw ore, and from other gangue and remilled three times. The flowsheet for concentrator operations in use in February 1946 is shown in Fig. 4. The flotation concentrate goes to a Dorr thickener and a drum filter. The drum filter, which delivers graphite with 20 pct moisture to the furnaces, is being replaced by a large centrifuge which delivers graphite with only 5 to 6 pct water, giving a substantial fuel saving. The dried furnace product aver-

Table III. Kropfmuehl Requirements

	Annual Requirements	Requirement per 100 Tons of Graphite Concentrate
Coke, tons	1100	18
Bituminous coal, tons	240	4
Lignite briquettes, tons	120	2
Diesel oil, gal	1600	26
Lubricants, tons	12	0.2
Dynamite, tons	18	0.3
Carbide, tons	9.6	0.16
Flotation reagents, tons	21	0.35
Drill steel, tons	8.4	0.14
Drill steel bits, No.	720	12
Pipe, ft	4000	66
Mine timber, cu m	840	14
Paper sacks, No.	120,000	2000

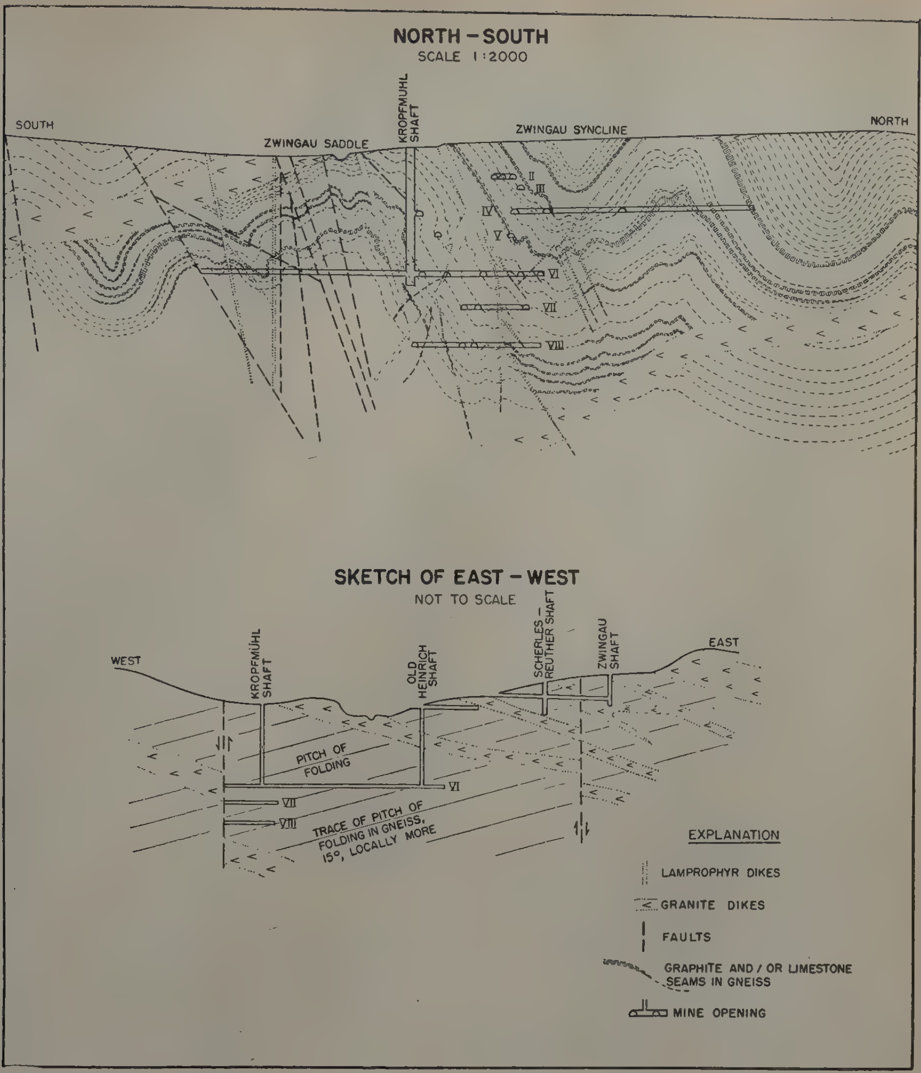

Fig. 2—Kropfmuehl structure sections.
After E. Hartmann.

ages 88 to 92 pct carbon with the balance mainly silicates.

Passing through a dust collecting system, the dried graphite enters 10 batteries of vibrating screens ranging from coarse down to one that retains particles 0.015 mm in size. The screened fines go directly to bagging or are combined in mixers to meet customer specifications. Most screened fines are medium to low in grade, containing from about 90 pct down to as low as 75 pct carbon. Lower qualities down to 40 pct are produced on demand.

The flake graphite is milled and rescreened four times to increase its purity and to separate broken crystals. The seven Pallmann mills made in the Saar for this purpose are 4-ft diam granite disks, the top disk stationary, the under disk rotating at about 5 rps on a vertical axis. For customers like the pencil companies requiring a high-purity, very fine graphite powder, some flake graphite is pulverized in hammer mills. Laboratory control at all stages appears to be good. The average analyses of several of the better qualities of product in September 1948 are given in Table I. These qualities are also notably free of mica.

The Kropfmuehl mill gets about 94 pct recovery on mill feed running 20 to 25 pct graphite. It is designed to treat about 136 metric tons per day, and in 1944 it actually reached this rate. Today it treats about 100 metric tons. The graphite concentrate output capacity is about 1.0 to 1.3 metric tph. In 1944 the average was 30 metric tons of concentrate per day and the present output is about 25 metric tons per day. Labor employed for milling and other surface work normally totals about 180 men. The ratio of concentrate to ore was 1:5.0 in 1938 and was increased to about 1:4.1 by 1944.

Production Statistics

The earliest statistics available show a production of 798 metric tons in 1868 in the Passau area. Crude ore production rose from 5000 in 1908 to around 41,000 metric tons by 1918, and then dropped off to around 10,000 to 15,000 metric tons per year in the 1920's. In 1928 there were 11 mines with 14 shafts which together produced 17,500 metric tons of raw graphite worth about a half million Reichsmarks. By 1935 Kropfmuehl dominated the field, and their production in 1938 amounted to 28,100 metric tons of crude graphite from which about 7700 metric tons of graphite concentrate were produced containing 5730 metric tons of carbon. In 1942 production started to climb rapidly from the 1938 level, and in 1944 it amounted to 36,400 metric tons of crude and about 8900 metric tons of carbon in the concentrate. With the occupation, production stopped until March 1946. Kropfmuehl made large shipments and reduced its stocks prior to currency reform in June 1948 and then cut back its production temporarily

Fig. 3—Kropfmuehl mill.

because of a reduced inland market which has now recovered, see Table II.

Kropfmuehl requirements of selected mine supplies are given in Table III.

The electric power requirements of Kropfmuehl are at present about 300,000 kw-hr per month, and were as high as 450,000 kw-hr per month in 1944. The mine owns two hydroelectric plants with a combined capacity of 215,000 kw-hr per month when the water supply is sufficient. The balance is drawn from the Bavarian power grid. During power crises, such as that of the winter of 1946, the mine and the screening plant are kept operating but the concentrator is shut down part time. If the crude ore is stored too long awaiting concentration, the pyrrhotite and pyrite oxidize give bad flotation efficiencies and a high iron oxide content to the graphite.

There was no war damage at Kropfmuehl, only some pilferage of roofing.

Passau graphite has been well known in Europe since at least the beginning of the 15th century. Crucibles used by the alchemists were made of it, and Agricola mentions them for their refractoriness. The first crucible manufacturing was apparently at the village of Hafnerzell (now called Obernzell), and the clay bond was also mined in the vicinity. Crucibles of Passau graphite had a world market until about 1810 when Ceylon graphite displaced it, later itself to be displaced in many countries by Madagascar graphite. But again during World War I the Passau graphite had to cover almost all of the crucible needs of the Central Powers. After this war, the Weimar Republic government attempted to reduce imports and encouraged the continued use of Passau graphite for crucible purposes by requiring at least 50 pct Passau graphite in the graphite portion of the crucible formula. This requirement was lifted in 1922, however, partly because of quality. Under the Nazis there was an importation of about 1200 tons of Ceylon graphite and 900 tons of Madagascar graphite during the average year of the mid 1930s. Later with the completion of the modern flotation plant the quality was improved, and the prejudice of the German and other European crucible manufacturers against Passau graphite has now largely disappeared. In the period of the present occupation, crucibles are being made without imported graphite and the Kropfmuehl Co. claims that it can meet all quality requirements of its customers.

From the German viewpoint, there is no particular reason to emphasize the importance of flake sizes. The fine sizes also satisfy industrial needs for which imports would otherwise be necessary. In the export market, neighboring countries are interested in the fines as well as the flake, and they pay prices depending more on the carbon content than on the grain size. They are particularly interested in the uniform standard quality of deliveries.

The present grades and qualities produced by Graphitwerk Kropfmuehl are given in Table IV, which shows the carbon analysis, screen size, the internal German controlled price as it was prior to currency reform, and the monthly production and probable export availability of each grade. It will be noticed that the AF grade and three other grades of fines were valued equal to or higher than the flake graphite. It should be borne in mind, however, that price control on graphite has now been eliminated in the German market, and graphite for export is sold at competitive world prices.

Large industrial consumers use the Passau graphite for crucibles, pencil leads, lubricants, carbon brushes, search light carbons, electrodes for electric furnaces and for batteries, foundry facing, polish, pigment, etc., see Table V. About 25 pct of the

Table IV. Kropfmuehl Graphite Products

Grade Symbol	Carbon, Pct	Screen Size[a]	Maximum German Price, at Mine, Bagged, RM[b]	Kropfmuehl Monthly Production Estimate, Metric Tons		Description of the Graphite Product
				Total	Available for Export	
Flake Graphite:						
S 40	93/95	91/40	430	45	15	Large flake for crucibles
NFL	93/95	85/40	388	180	60	Normal German crucible flake
Graphite Fines:						
A F	96/98	1/140	640	55	25	Pencil grade, hammer-milled from flake
EDM	96/98	N.A.	493			Sometimes pencil grade, very fine
F P	96/98	24/100	430			Fine
EDM	90/92	3/140	400	25	15	Very fine
F P	90/92	N.A.	367			Fine
F P	85/87	33/100	304	100	40	Fine
F P	83/85	N.A.	278	75	30	Fine
F P	75/83	N.A.	225	20		Fine
		Total Production		500	185	

[a] Pct on a screen of given size: e.g. 33/100 means 33 pct on screen with 100 holes per sq cm.
[b] Prior to currency reform, June 1948.
For comparison:
40 holes per sq. cm. corresponds approximately to 18-mesh, U.S. Standard, or 1.0 mm particle size; 100 holes to 30-mesh, 0.6 mm particle size; and 140 holes to 35-mesh, 0.5 mm particle size.

Kropfmuehl graphite, however, is sold to dealers and its end use is therefore difficult to trace. Several large customers are located in what is now the Russian Zone of Germany. They include Lorenz Werk at Hainsberg and Graphit-Schmelztiegelwerk at Dresden for the manufacture of crucibles, and three companies manufacturing dry cells, two of which are in the Russian Sector of Berlin.

To summarize the viewpoint of the producers of Passau graphite with respect to their fines and powdered products, they would emphasize the uniform delivered size and quality of the various grades, also the low density, the very fine particle sizes of selected powdered products, the high refractoriness, and the good heat conductivity. They claim the density to be 20 pct lower than that of competitive materials which would result in a saving in certain applications, for example, of up to 25 pct in mixtures of the finest powders with oils for paints or lubricants.

The largest pre-war customers for flake graphite were the United States, Italy, England and Czechoslovakia; while the customers for Passau fines included almost every country in Europe, see Table VI. Contracts in the post-war period have been concluded with Czechoslovakia, Italy, England, Holland, and other countries at prices comparable to Madagascar graphite cif. European ports.

Graphite Crucible Manufacture

Crucibles were first manufactured in Germany in the Passau area. Probably because of insufficient clays of suitable properties in the Passau area and of difficulties with coal supply, the crucible manufacturing industry moved to the northwest and settled in Hesse and Thuringia. The Grossalmerode area, which is well known for its clay, is now the center of the industry in Western Germany. In this area there is also the manufacture of nongraphite crucibles, including glass pots.

It is the practice in Germany to specify that both the graphite and the clay must be free of impurities such as sulphur, iron oxide, and lime. The mix ranges upward from 30 pct graphite, and has 0 to 25 pct quartz with the clay, depending upon the intended application of the crucible. Some clay from Klingenberg, Bavaria, is added to the local Grossalmerode clay to increase its plasticity. The process used has been appraised by five British ceramists[7] who visited the area in 1945 with the purpose of observing any new methods. They concluded that the quality of product was to be attributed more to the care and craftsmanship of the workers than to modern practice or to any excellence in the plant equipment or its technical control. The constituents are mixed in Eirich mixers, then pugged and cut into lengths for maturing. The matured material is repugged and then fashioned into large balls by handwork. The crucibles are shaped from the balls by jolleying in plaster molds. After up to five weeks of drying in steam-heated drying rooms, the warm crucibles are painted with two or three coats of pottery-type glazes and fired in periodic kilns of a down-draft type at temperatures ranging from 1200°C to about 1350°C.

Since Madagascar flake graphite is at present the basis of the American graphite crucible industry, the properties and specifications for crucible graphite in America as reported by Gwinn[8] are analyzed here and compared with the properties and specifications of Passau crucible graphite. As delivered, it is apparently higher in carbon content and lower in free impurities, especially mica, than the Madagascar graphite. These facts are evidently considered important in Germany, although Gwinn points out that American manufacturers seem to believe that there is nothing to be gained by using graphite containing more than 85 pct carbon since higher carbon content would mean reduced toughness of the flake.

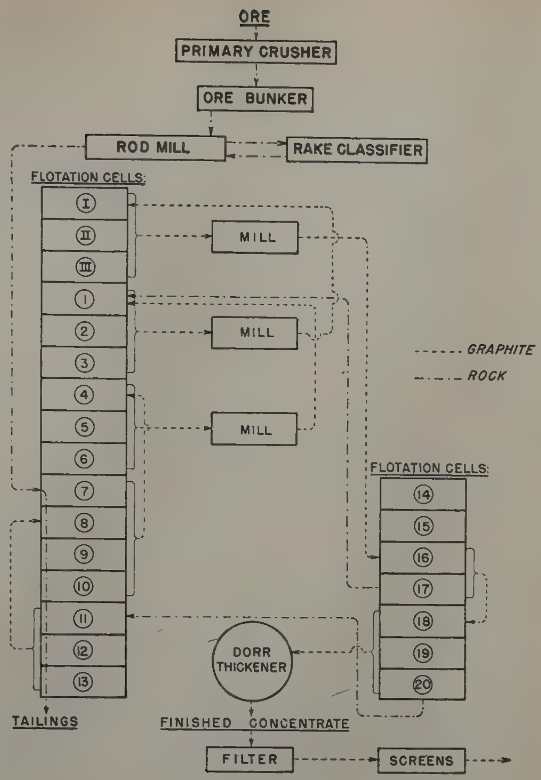

Fig. 4—Graphite concentrator.

As for size of crystals, the Madagascar flake can probably be delivered in commercial quantities in larger size flake than can the Passau graphite. Under the microscope the Passau S 40 grade appeared to average about 0.7 mm in diam and to range from 2.0 mm down to less than 0.5 mm. An available specimen of Madagascar graphite also examined showed about the same size range but a larger average diameter, and the crystals were thicker. It should be noted that Graphitwerk Kropfmuehl has stated that it could make a small tonnage of a coarser, S 20, grade on demand, which would probably average about 1 mm in diam.

Table V. West German Major Consumers of Passau Graphite

Firm	Town	District	Qualities
For crucibles:			
Becker and Piscantor	Grossalmerode	Hesse	S 40 and NFL
Aug. Gundlach	Grossalmerode	Hesse	S 40 and NFL
Carl Nolte	Frankenhain	Hesse	S 40 and NFL
Cosack and Co.	Neheim	Ruhr	NFL
For pencil leads:			
A. W. Faber-Castell	Nürnberg	Bavaria	AF
Eberhard Faber	Neumarkt	Bavaria	AF
J. S. Staedtler, Mars	Nürnberg	Bavaria	AF
Schwan-Bleistift	Nürnberg	Bavaria	AF
Joh. Froescheis, Lyra	Nürnberg	Bavaria	AF
For dry cells:			
Bavaria G.m.b.H.	Munich	Bavaria	FP 85/87 pct C.
Pertrix	Hannover	Lower Saxony	FP 85/87 pct C.
Koch and Krueger	Neukölln	Berlin	FP 85/87 pct C.
Schmidt and Co.	Charlottenburg	Berlin	FP 85/87 pct C.
Berliner Batterie	SW 61	Berlin	FP 85/87 pct C.

Table VI. Prewar Kropfmuehl Export Pattern, Metric tons per Year

	Flake			
	S 40	NFL	Fines	Total
Austria	5	10	130	145
Belgium			30	30
Czechoslovakia	30	80	320	430
Denmark			40	40
England		150	650	800
France			80	80
Holland			60	60
Hungary			100	100
Italy	120	290	310	720
Jugoslavia			40	40
Poland	35	30	50	115
Rumania			70	70
Spain			115	115
Sweden			120	120
Switzerland		15	85	100
United States	70	150	20	240
TOTAL	260	725	2220	3205

Compared to the Passau graphite, the Madagascar graphite showed relatively smooth surfaces and smooth edges that were either round or straight. On the principal cleavage surfaces of both there appear to be traces of three cross cleavages at approximately 60° angles, but in the Madagascar specimen they were more conspicuous and, in many crystals, parallel to the straight edges. It would appear that the Madagascar graphite specimen could have been better sized than the Passau graphite because of its smooth edges but actually it was not. Under the microscope it was noted incidentally that the Passau graphite has a higher lustre and reflection.

No specific information was available on Passau graphite with respect to the toughness or the packed-volume properties referred to by Gwinn.[8] However, it was claimed for the Passau graphite in the Tonindustrie-Zeitung[9] that despite its smaller flake size compared with Madagascar graphite, the Bavarian flake graphite is better for crucible manufacture because of its two superior properties, its superior refractoriness and its high heat conductivity. It is recognized, however, that American practice sometimes shows that a more refractory graphite does not necessarily give a better service life since crucibles, except those for aluminum, usually fail from abuse rather than from burning out.

Because of its flake shape, the Bavarian graphite enjoys the same advantage noted by Gwinn[8] as the Madagascar graphite over an acicular or granular graphite in that it can be oriented to form an interlocking structure with the flakes parallel to the wall surface of the crucible, thus providing the flexibility and strength necessary to resist the great thermal shocks to which crucibles are subjected. It is also claimed for the Passau crucible graphite that the crucible manufacturer does not have to clean it or mill it further.

Compared to American domestic flake graphite and again referring to the report by Gwinn,[8] it would appear that the Passau graphite would have had an advantage over the wartime domestic flake graphite in sizing and in toughness due to the probable superiority of the processing in the Kropfmuehl mill. It may be assumed that the flake toughness of the finished Bavarian graphite would have been greater because of its milling and given less breakdown in the mixing process of crucible manufacture. Better grading perhaps would have reduced the packed volume and therefore the necessary amount of clay binder.

It was learned by correspondence that the German crucible manufacturers do not anticipate any volume of crucible export to America at present, one reason being the 20 pct ad valorem import duty. The 30 pct ad valorem import duty on flake graphite itself should not, however, restrict direct import of German graphite any more than any other foreign graphite.

Acknowledgments

The material for this paper is drawn from observations made and information collected at the Kropfmuehl mine and concentrator and at their Munich offices during inspections in December 1946 and November 1948. In the latter inspection the writer was accompanied by D. E. Hartmann of the Geological Dept. of the Bavarian Bureau of Mines, whose work is drawn upon for the sketch map and structure sections, shown in Figs. 1 and 2. Pertinent Military Government reports also were used, and the standard published references consulted are listed in the bibliography.

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Rock Hardness as a Factor In Drilling Problems

by W. B. Mather

Literature dealing with rock drilling presents a mass of conflicting data. The principal cause of the confusion is attributed to varying definitions of the hardness factor of rock and cutting media. The future development of more efficient drilling methods demands that this problem be solved. The first step in the solution is the compilation of complete data on the physical properties of rocks.

A SURVEY of the technical literature concerned with oil well drilling methods and particularly with rate of penetration by various cutting media on different types of rock provides a mass of conflicting data. This is especially true with respect to, (1) the relative efficiency of diamond, rotary and percussion drilling operations; (2) the costs of the various types of drilling; and (3) the hardness factor of various types of rock and cutting media. The latter is the particular phase of the subject discussed here.

The cause of the confusion may be attributed partially to a lack of basic knowledge concerning the physical properties of the cutting media and of individual types of rock. It is also due in part to a general lack of uniform standards in the industry for evaluating the hardness versus drillability of rocks while possible errors in interpretation by the authors of the various papers pertaining to the subject may be an additional cause of misunderstanding.

The hardness of a mineral is defined by Dana[1] as "the resistance which a smooth surface offers to abrasion." Webster[2] states that hardness is "the cohesion of the particles on the surface of a body (as a mineral) as determined by its capacity to scratch another or be itself scratched."

Rock hardness, although not usually defined in publications describing rocks, is assumed to be the same as the minerals which compose the rock or the resistance of the rock to abrasion.

Drillability of a rock may be defined arbitrarily as the rate at which rock is penetrated by a definite type of drilling tool with the assistance of a specific type of cutting surface or medium and under specified conditions as to speed of rotation and applications of pressure.

Eventually the term penetrability may have to be employed to include any method of making boreholes in rocks, since methods other than those using conventional cutting surfaces are already in operation. A case in point is the jet piercing method[3] which employs an oxy-acetylene flame as a means of penetration.

The word hardness as used by the drilling fraternity has a variety of meanings which depend upon the type of drilling method employed. For example, in diamond drilling, hardness is interpretated as resistance to abrasion. In percussion drilling the term implies resistance to impact or indentation, while in rotary drilling, hardness is considered analagous to compressive strength. The three types of hardness—abrasion, impact and compression—are conflicting variables for different types of rock. Nevertheless, the term hardness is applied indiscriminately by the industry for the resistance of rock to penetration by any type of drilling technique.

Abrasion Hardness

Since 1818, Moh's scale of hardness, although only relative in nature, has been accepted universally as the method of measuring mineral and therefore rock hardness. In 1933, Ridgway, Ballard and

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Bailey[4] modified the upper, or harder, portion of Moh's scale, as shown in Table I.

If metallic media were included in the extension of Moh's scale, stellite would be placed at 8 and tungsten carbide at 12.

In 1935, Wooddell[5] measured the relative resistance of minerals to abrasion during lapping and proposed further refinements to Moh's scale in an effort to reduce the relative character of the scale and provide a numerical hardness value more in keeping with the comparative resistance to abrasion of the various minerals. This vastly improved measurement of abrasion hardness is shown in Table II with two sets of values depending on the choice of two mineral standards in Moh's scale.

In other words, the hardness ratio of South American brown bortz and quartz is 6:1 instead of 10:7 as one would infer from the old Moh's scale. The low tensile strength of diamonds permits their use only in diamond drilling where the load on the bit is relatively small and impact is reduced to a minimum. However, in the case of silicon carbide and fused alumina, which also have low tensile strengths, their lower hardness values does not permit their economical use in the facing of drill bits. After diamonds, boron carbide is the hardest mineral substance, and although it is very resistant to abrasion, its low tensile strength prevents its use as a cutting medium.[6]

Tungsten carbide, because of its greater tensile strength and a hardness value of nearly twice that of quartz, has been found to be the most satisfactory material for the facing of rotary and percussion bits where load and impact factors are of major importance.

In diamond drilling operations in Crane County, Texas, McCray[7] reports, "As different strata of limestone drill at widely different rates, varying from perhaps 6 or 8 to 90 minutes per foot, and as sandstones generally core at rates from 5 to 10 minutes per foot, much faster on the average than limestones or dolomites, it must be concluded that hardness is not a factor of the rate of penetration." He also observed that "very finely crystalline limestones were the slowest drilling of all rock sections penetrated. Limestones and dolomites formed of larger crystals drilled faster—sandstones which drill faster than limestones, in general have grains whose sizes are considerably greater than limestone crystals. On the other hand, the mineral of finest crystalline structure, the chert, is drilled at rates from 20 to 30 minutes per foot or 3 to 5 times faster than the finely crystalline limestone. This fact is largely explained by the tendency of chert to chip out with a characteristic conchoidal fracture."

According to Christensen's charts[8] of diamond drilling time on the various members of the Weber sand section at Rangely Field, Colo., dolomite was penetrated at an average of 8.5 min per ft, sandstone at 8.7 min per ft and shale at 13.4 min per ft.

Table I. Modification of the Upper Portion of Moh's Scale

Moh's Scale	Extension
(6) Orthoclase	(6) Orthoclase or Periclase
	(7) Vitreous Pure Silica
(7) Quartz	(8) Quartz
	(9) Garnet
(8) Topaz	(10) Topaz
	(11) Fused Zirconia
(9) Sapphire	(12) Fused Alumina
	(13) Silicon Carbide
(10) Diamond	(14) Boron Carbide
	(15) Diamond

Table II. Measurement of Abrasion Hardness

	Scale	
	Corundum=9 Diamond=10	Quartz=7 Corundum=9
Rock crystal quartz	8.94	7.0
African crystal corundum	9.00	9.0
Fused alumina	9.03	10.0
Fused alumina (3.14 pct TiO_2)	9.06	11.0
Tungsten carbide (13 pct cobalt)	9.09	12.0
Green silicon carbide	9.13	13.4
Black silicon carbide	9.15	14.0
Boron carbide	9.32	19.7
South American carbonados	9.82	36.4
Belge Congo gray opaque (cubic xals)	9.89	38.7
Belge Congo clear white (cubic xals)	9.95	40.7
Belge Congo yellow (cubic xals)	9.96	41.0
South American ballas	9.99	42.0
South American brown bortz	10.00	42.4

From experience in diamond drilling pre-Cambrian marbles, dolomites, and quartzites, the author observed that the marbles and dolomites were penetrated much more rapidly than quartzites. These observations indicate that the resistance to abrasion of the respective mineral components of the rock was definitely a factor in rate of penetration. On the other hand, the greater difficulty often experienced in diamond drilling shale is incongruous, since theoretically it is composed of much softer minerals than those in limestones, dolomites, and sandstones.

In Southwest Research Institute drilling experiments using tungsten carbide, alundum and carborundum as cutting media, it was observed that fine-grained Austin chalk was penetrated much more rapidly than coarser grained Edwards limestone. This fact is explained partially by the greater degree of cementation in Edwards limestone than in the Austin chalk in spite of the fact that chemical or mineralogical analyses would show that both rocks had nearly identical compositions.

Impact Hardness

Resistance to impact or indentation is the principal type of hardness envolved in percussion drilling. Klapka[9] presented the data shown in Table III based on the indentation tests of Knoop, Peters, and Emerson of the National Bureau of Standards and Wooddell's abrasion tests previously mentioned, as compared to Moh's scale of hardness.

The variance in the comparative hardness of the cutting media by the two methods of measurement are obvious, where the ratio of diamond to tungsten carbide and to quartz by the abrasion test is 3.5:1 and 6:1. By the indentation method, it is more than 7:1 and more than 10:1. More conspicuous, however, is the reversal of position of tungsten carbide and corundum in the two tests with tungsten carbide, the harder material by abrasion tests, and corundum, the harder by the indentation method.

Irrespective of these laboratory tests, in actual drilling practice tungsten carbide is used in facing percussion tools because of its toughness and ability to withstand impact. The materials shown to be most resistant to impact by the indentation tests, such as diamonds, silicon carbides, and corundum, cannot be used because of their ease of fracture under impact. It therefore appears that in measur-

Table III. Moh's Scale Compared with Indentation and Abrasion Tests

Indentation Tests	Moh's	Abrasion Tests	Moh's
Diamond 8200-8500	10	Diamond 42	10
		Black silicon carbides 14	
		Tungsten carbide 12	
Silicon carbides 2050-2140			
Corundum (sapphire) 1700-2200	9		
		Corundum 9	9
		Quartz 7	7
Tungsten carbide 1050-1500			
Topaz 1250	8		
Crystalline quartz 710-790	7		
Gypsum 32	2		
Talc	1		

ing impact hardness the elasticity and tensile strength of the material are the important factors which should be determined.

Compression Hardness

Table IV by Norton Co.[10] of resistance to compression (compressive hardness) indicates that this type of hardness as applied to cutting media compares favorably with abrasion hardness and to a lesser extent with impact hardness.

The ratio of hardness of silicon carbide to quartz by the three methods is: abrasion hardness, 2:1; impact hardness, 2.7:1; compression hardness, 1.4:1.

Scott[11] in discussing hard-rock rotary drilling, defines hardness of a rock as "its comparative crushing strength or resistance to penetration." He further defines drillability "as a rate-weight ratio where rate is feet per hour and weight is pounds force acting on the bit." These definitions, which apply only to penetration of rock by rotary drilling tools, are in agreement with other authorities who believe that hardness in rotary drilling is resistance to compression or crushing and therefore the compressive strength of a rock would be a measure of its hardness.

The relation between compression and impact hardness as demonstrated by Bond[12] are given in Table V.

This table clearly exemplifies the discrepancies existing between compression and impact hardness. For example, according to the compression tests, shale is the softest rock, while Virginia granite is softer than six limestone specimens but harder than the remaining three limestones. On the other hand, according to impact hardness, shale is harder than granite and in addition is harder than seven out of nine limestone specimens. If abrasion tests had also been performed on these specimens, the results would have followed more closely the results of the compression rather than impact tests except that the two granites, which by definition are composed mostly of quartz and feldspar, would have proven harder than all limestone specimens, with the shale the softest of the materials tested.

Summary

Only the most obvious inconsistencies in our present usage of the term "hardness" have been presented.

In the case of diamond drilling or in similar methods of rock penetration where abrasion is believed to be the governing factor, there are other variables which cloud the picture. For example, there is the contention that finer grained rocks are more difficult to penetrate than coarser grained rocks of identical mineral composition, where the only apparent difference in these two rocks is that the finer grained type contains more grain boundaries than the other, the inference being that grain boundaries exhibit a greater resistance to abrasion than the interior of the grain. In addition to rock texture, the degree of cementation, variation in mineral content, and existence of incipient fractures also have a marked effect on abrasion hardness. The longer drilling times required to cut shale[13] than dolomite and other rocks of greater abrasive hardness may be attributed to the lubricating action of the soft shale particles on the drill face. The very fact that in certain cases chert, a rock of highest abrasive hardness, cuts more rapidly with a diamond drill than fine-grained limestone[14] indicates that other physical properties of rock are of major importance. The effect on abrasion hardness of other physical properties, such as elasticity, thermal expansion, grain and crystal orientation, compressive and tensile strength, etc., have not been definitely determined. However, Norton[15] states that hardness or resistance to abrasion is a function of interatomic distance, where the hardness decreases with increase in distance between the atoms if the charge of the ions is constant. Since the interatomic distance is dependent on the ionic charge, both factors are at least partially responsible for this type of hardness. He further points out that compounds with layer structures are usually softer than those without the layer structures and that hardness may vary with crystallographic direction because of crystal structure. This latter factor he believes to be of less importance than interatomic distance and ionic charge.

The present value of impact or indentation hardness and tests as a measuring device for ease of penetration by percussion tools is highly questionable. It appears that elasticity, tensile and compressive strength are at least in part the governing factors in resistance to impact. The observation that percussion tools are often more successful in drilling shales than rotary and diamond drills does not depend upon the impact hardness of the shale, which is very high while its compression and abrasion hardness are low, but rather that the lubricating action of shale particles has little if any effect under impact while they are effective on rotating bits.

Table IV. Resistance to Compressive Hardness

Material	Ultimate Strength in Compression (psi)
Molded boron carbide	300,000
Dense silicon carbide	82,000
Crystalline alumina	75,000
Optical quartz	58,000
Fused silica	38.000

Table V. Relation Between Compression and Impact Hardness

Material	State	Compression, Psi		Impact, Ft-lb	
		Max.	Min.	Max.	Avg.
Limestone (fine)	Mich.	15,450	14,220	12.0	6.09
Limestone (white)	Mo.	10,830	10,285	7.8	6.28
Limestone	Mo.	18,120	16,100	12.1	7.18
Granite	Va.	17,200	15,020	13.4	11.20
Granite	Ga.	27,600	17,923	14.8	11.42
Limestone	Wash.	22,000	16,685	17.5	12.20
Limestone	Wyo.	30,000	27,750	20.5	12.60
Limestone (gray)	Mo.	15,960	14,410	19.7	13.16
Limestone (coarse)	Mich.	19,100	16,785	27.0	14.75
Shale	Pa.	9,100	8,160	22.1	15.06
Limestone	Ark.	24,480	20,000	33.2	17.91
Limestone	Mo.	23,540	19,222	23.8	18.00

As previously indicated, compressive strength may be an important factor in diamond and percussion drilling. Rotary drilling of granite is recognized by the industry as being more difficult than the drilling of limestone, yet according to compressive strength tests, Virginia granite should be easier to penetrate than six of the nine limestones tested by Bond,[16] while Georgia granite should be less difficult to drill than three of them. Therefore, the suggestion that compressive strength is the governing factor in rotary drilling is open to question since other factors such as elasticity, abrasion hardness, and tensile strength are believed to be equally contributing factors.

The tensile strength and elasticity of minerals and rocks as well as cutting media are believed to be two of the most important factors in drilling of any type, yet they are almost totally ignored in studies of drilling problems.

Other physical properties of rocks, at present considered unimportant, may be the bases of developing new drilling techniques and equipment in the future; as an example, in the jet piercing method of drilling, where thermal expansion is the important physical property.[17]

The disparity between laboratory data and actual drilling results may be attributed partially to variations in the physical properties of minerals and rocks when buried under several thousand feet of rock. If this is the case, the question then arises as to the character of the changes and the rate of variations likely to be encountered at various depths below the earth's surface.

Conclusion

The need for more complete data on the physical properties of rocks is apparent. A considerable portion of the information that may be found in the literature should be compiled and a research program instituted to furnish the missing portions in the data.

Since this study was undertaken on behalf of petroleum industry, the data were to a large extent secured from that source. In spite of the fact that in the past the diamond drill has been primarily the tool of the mining industry, apparently the oil well driller has to a large extent ignored the experiences of the miner in the use of this tool. The petroleum industry has attempted to equip the rotary rig with diamond tools for which they were never designed. The oil well driller has learned the hard way that the slow rotary speeds and high bit pressures necessary for rotary drilling will not produce satisfactory results with a diamond bit. Similarly if the mining industry anticipates the use of rotary and cable-tool drills, it would be wise to study the experiences of the petroleum industry.

The basic problem of determining the physical properties of rocks is applicable to both industries and the knowledge gained will be of equal benefit to both. Therefore I believe that the work of Obert, Windes and Duvall at the U. S. Bureau of Mines[18-21] on the determination of the physical properties of mine rock could be employed as a starting point for future investigations. Their excellent work included tests of sonic properties, compressive strength, modulus of rupture, scleroscope and abrasive hardness, and impact toughness on various types of mine rock. Their suggestion that detailed petrographic studies should accompany the physical tests is sound. They also found that there are few studies in the literature pertaining to the hardness of rock.

Since complete information on the physical properties of rock should be of greatest potential value to the petroleum and mining industries, it would appear that they are the logical sponsors of such a research program.

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A Pattern for Sound Fuel Procurement

by Marshall Pease, Jr. and Raymond J. Brandon

A pattern for providing a large utility, The Detroit Edison Co., with an adequate fuel supply is outlined. From the standpoint of both fuel procurement and utilization, consideration has been given competition relationships, fuel properties as well as operating conditions that must be met to provide dependable service and reliable plant operation.

A UTILITY that has a large consumption of coal must insure an adequate and sound supply of fuel. The Detroit Edison Co., which has an annual coal consumption of about four million tons and spends approximately $32 million a year for coal, including freight charges, has developed a program for fuel procurement and for evaluation and selection of fuel for reliable and efficient plant operation.

Fuel Procurement

Coal Purchasing Division: The Fuel Supply Div. of the Purchasing Dept. combines all of the procurement functions in one group, which must maintain adequate stocks of fuel. In addition to its usual purchasing duties, the division also governs transportation, follow-up, invoice and freight bill, in cooperation with the Accounting Dept. The Fuel Div. is comprised of the fuel agent, an assistant fuel agent, a coal buyer and five coal clerks, who follow the movement of each car, initially approve freight bills and invoices, and file claims whenever there is a shortage of one ton or more. The fuel agent reports directly to the purchasing agent of the Company, and all major programs are planned jointly with the chief purchasing officer.

The apparent individuality of the fuel section is necessary because of the tremendous volume of coal cars handled each day, sometimes as many as 500, which must be handled promptly to complete the fastest possible move from the mines to the plants.

Determining Annual Coal Volume: Through the combined studies of a Production Dept. Load Committee and the Controller's office, accurate predictions of coal requirements for any given year are provided the Fuel Dept. from 12 to 15 months in advance. This is divided into the requirements of all individual power plants and central heating plants. This in turn determines the quantity and quality for plants whose specific fuels vary with the type of equipment installed. In the Detroit area, which has a high industrial load, early estimates of annual coal consumption usually are resolved at the end of the year within 4 or 5 pct of the original plan.

Operating in a highly developed industrial area eases the task of estimating primary output; and, with the residential demands increasing in a steady and fairly well-defined pattern, the overall coal schedules are not subject to radical changes during any given year.

Selecting Suppliers: At any time, but especially during or before an emergency, the coal supply factor must be made secure. This is particularly true in the procurement of utility fuel. There are always extreme quantities of so-called "bargain" coal available during the buyers' markets, such as recently prevailed. These opportunistic offerings may be considered a means of averaging down overall price, rather than as a steady and dependable supply source.

Coal is purchased on contract from mine operators and sales agents who have proved reliable. They are not opportunists who desert for higher dollars in times of duress; they do not fail to fulfill contracts when markets rise or overship when markets dip.

The progressive operator today who is willing to expend capital to improve quality and service deserves much more consideration than a matching of short-term pennies.

When a company has a high volume of annual needs, all phases of the mine supply must be considered. The mine must be able to produce the quality required at a fair price and be able to sell oversizes of coal in enough volume to screen sufficient nut and slack. It must crush coal and sell mine run at prices in line with competitive nut and slack and be willing to do so when there is no demand for prepared sizes.

Determining Price: The public utility is in direct competition with any of its customers who can, if savings are guaranteed, generate their own power. Today, to a greater extent than ever before, private industry must do a better overall job than Government-controlled operations. The price of coal must be realistically balanced with the price paid by almost all industries and the railroads as well.

The supply-demand ratio in coal is not difficult to discern. There is access at all times to Govern-

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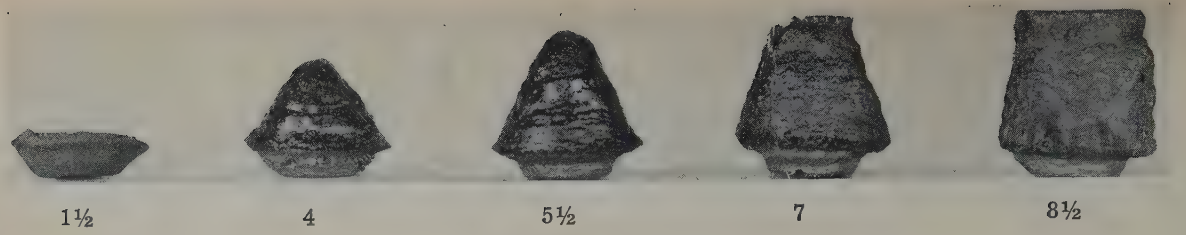


Fig. 1—Coke buttons obtained by British standard method for the crucible swelling test for coal.

ment Bureau indices, to car-loading records, to weekly production figures, and to consumption of coals by all users. Continuing studies of these phases will give the overall outlook in a broad sense. Actual market prices by reviewing government, state and municipal bids must be checked and the market followed daily through personal contacts.

When coal production is greater than the demand, there is ordinarily a price spread in favor of open market purchase over contract purchase. Here, the utility buyer must try to forecast the longer term prospects and adjust his ideas of the necessary contract tonnage protection to the overall economics. The Detroit Edison Co. uses mutually acceptable cancellation clauses on its coal contracts, which give both parties the right to arbitrate the price when lower market prices justify. A sensible balance must be maintained between price, quality, and dependability of supply at all times.

Proper Storages: All of the company's plants are located so that a water-borne self-unloading boat delivery from Lake Erie loading ports has proved to be the most economical and practicable method of delivery. Hence, nearly all annual coal requirement is floated during the eight to nine months of navigation. In this respect, sufficient coal is delivered by boat to insure full consumption for the entire system between March 15 and Dec. 1, plus enough storage coal to provide about 150 days' supply at Dec. 1.

If coal is bull-dozed into all power plant storage sites, there is no danger of spontaneous combustion. Weather conditions, therefore, do not delay stocking programs, and coal is put on the ground during each and all of the months of navigation.

The Fuel Div. works closely with the Production Dept. in advance of each month's coal delivery to plants. Schedules are arranged to provide the most flexible means of boat delivery as well as reduce to a minimum any cause for overtime in the stocking of coal.

Market Studies: To properly define a coal market, the geological factors of coal origin and development must be known. This is accomplished through general tests and Bureau of Mines data. The coal buyer must know the characteristics of coal by seams, whether friable or firm, and of byproduct or steam quality.

Information on competing producing districts and the usual markets supplied may be obtained from the Dept. of Interior, Bureau of Mines Industry Surveys, and also coal trade association reports and studies.

The freight rate patterns on all coals that might be available for use must be at the buyer's disposal at all times. This requires a familiarity with railway tariffs and a close association with railroad coal department representatives. The fuel buying policies of railroads using coals similar to those of the purchaser can often assist in determining reasonable prices.

When competitive buying is a factor, the coal buying policies of other large consumers can be determined by monthly Dept. of Interior reports of consumption and storage, plus acquaintance, and the National Assn. of Purchasing Agents' monthly bulletins.

The conditions that control the market are national production, consumption, storage programs, seasonal fluctuations, the retail situation, and export needs. This knowledge is acquired by experience and continuous contacts with people involved in the entire coal industry.

Knowledge of the operating end of the mining business is important in fuel procurement. The buyer must know the type of mining, costs, cleaning facilities, capacity, life of mine and future development, financial condition and the mine management personnel. Thorough inspection visits are required periodically to key mines.

Contract negotiations between the mines and the miners' union must be followed closely. The major upsets to coal production brought about by strikes during the past ten years clearly indicate the importance of providing sound storage protection for coal-burning utilities.

The coal buyer will benefit by a careful review of his coal specifications. If they are restricted, he must find out why. Plant modernization or expansion must be preceded by and based upon a comprehensive study of all the low-cost coals available.

Selecting and Evaluating Fuels

It would be impractical to attempt to set up a rigid pattern for utility fuel procurement that would be economically valid or sound for the varied fuel procurement conditions encountered in the utility industry. Some principles that may have some general application in regards to fuel procurement as related to power plant operation and equipment may be established by a description of this phase of the fuel procurement pattern of The Detroit Edison Co.

The primary concern of the utility is to have its product available at all times for the service of its users, and the procuring of fuel for reliable service and efficient plant operation is an important part of this overall aim. Since coal constitutes from 97 to 98 pct of the total fuel burned in the Company's plants, it is logical that discussion should center first around the procedure for selection and evaluation of this fuel.

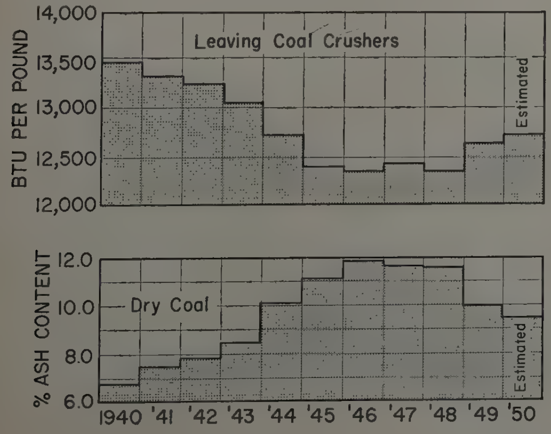
Coal selection is a matter of appraising a variety of combinations of chemical and physical properties of coal as they affect the performance of the plant in which the coal is to be burned. As an indication of the complexity of the problem, the properties of the coals may vary among individual mines, among producing areas and among seams. One or more of these properties may affect to some extent either the load-carrying capacity or operating expense of the plant. The limiting of steam generating capacity because of one or more of these characteristics is the most serious. If the plant cannot generate sufficient steam to carry the plant load because of some coal property, the coal having that property cannot be used at any price. The possibility of variation in coal properties, together with the changing competitive picture of fuels, makes it necessary to keep reviewing the fuel situation so that selection and evaluation will result in the most economical fuel from both a purchasing and operating standpoint.

Selecting Coal from One Field

In the selection and evaluation of coals from mines in one field or district, which is the most commonly made, the first step is the analysis of coal samples. To obtain these samples, two or more cars of coal are shipped direct from the mines to one of the company plants where representative samples are collected. As a means of checking the quality of the product from regular shippers, a testing schedule is set up whereby these coals are analyzed periodically. The number of times a mine is sampled during the year depends on the tonnage shipped and its variation in quality during past years. In considering the purchase and burning of coal from an unfamiliar mine in this field, the same procedure is followed, and after the samples are collected they are sent to the laboratory where they are analyzed for various chemical and physical properties. These properties are discussed from the standpoint of coal selection and their effect on plant capacity, efficiency, and operating expense.

Ash Fusion Temperatures: From an operating standpoint, the coal property that is of primary importance is the ash fusion temperatures. Since a large portion of our fuel-burning equipment is designed to burn coals with ash-softening temperatures of 2500°F or higher, coals with low ash-fusion temperatures can cause clinkering on the stokers and collection of ash deposits difficult to remove from the heat transfer surfaces of both stoker-fired and pulverized-fired furnaces. Besides causing continual operating difficulties and increased operating expense, these coals usually cause a decrease in boiler availability and capacity, resulting in reduction of plant load. For equipment designed to handle high fusion coals, selection should be limited to coals with this characteristic. Increased operating and maintenance expense may more than offset any delivered price advantage. The *ash-softening* temperature is a fairly reliable guide to the clinkering that can be expected for a coal fired on stokers, but the *ash-fusion* temperatures also must be considered in pulverized coal firing. The laboratory test for the ash-fusion temperatures is conducted in a reducing atmosphere that is similar to conditions surrounding the ash in stoker firing, but the fusion temperatures may be raised from 100° to 500°F if the atmosphere is oxidizing in the pulverized coal-fired furnace. In pulverized coal firing, the coal and ash particles are finely divided and if some of the ash constituents with a low melting point become separated from the ash, severe slagging may occur. In observing burning tests on coals on the borderline for this coal property, many coals performed satisfactorily until boiler ratings were increased or until furnace atmosphere conditions temporarily changed, allowing slag to start to collect on the tubes and becoming progressively worse until it became necessary to shut down the unit for cleaning.

Fig. 2—Average heating value and ash content of coal burned in power plants.



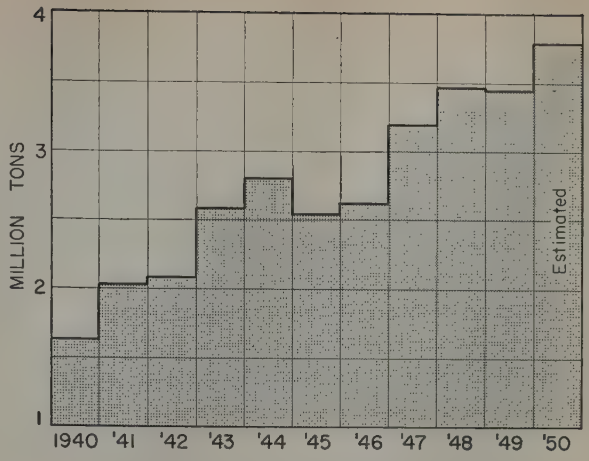


Fig. 3—Annual power plant coal consumption.

Moisture Content: Besides displacing useful heating value of the coal, high surface moisture values can cause difficulty in unloading coal from cars, plugging of coal-handling chutes and stoker hoppers and reduction in pulverizer mill capacity. Steam generator efficiency is decreased since the moisture in the coal must be evaporated and superheated.

Loss of feed to the pulverizer mills, which is caused by wet coal hanging up in the chutes and feeders, and loss of mill capacity due to high moisture values are probably the most serious of the operating difficulties where the direct firing system is installed. While the inherent moisture in the coal is not detrimental to coal preparation like other forms of moisture, it delays ignition and makes the fires unstable, particularly at low rates of firing. The moisture in coal first must be evaporated before the temperature of the coal can be raised to the ignition point, and the time required to evaporate the moisture causes the delay in ignition. If a coal with high moisture content is to be purchased and burned, consideration should be given to the means of supplying hotter air to the mill so that more of the moisture will be evaporated before the coal reaches the burner. Although the

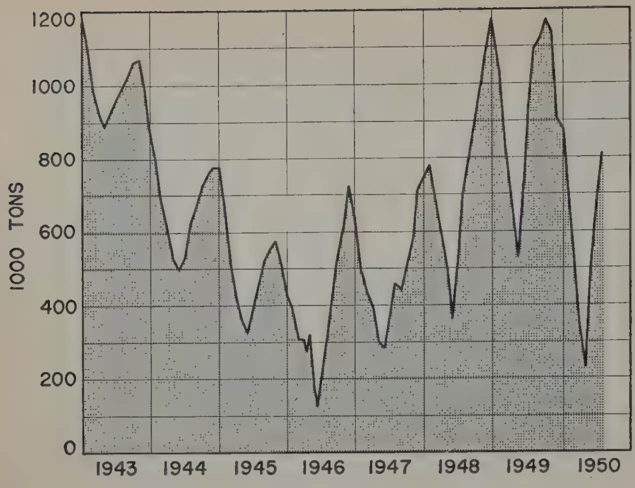


Fig. 4—Total power plant coal stock.

air temperature may be high enough at full load ratings, at low ratings it may not be enough to prevent the instability of firing that often occurs because of moisture. Coal feed to sections of the stoker may be stopped, resulting in parts of the grate area becoming bare with resultant increased maintenance expense due to replacing burned and warped iron.

Sulphur: The final flue gas temperature leaving the economizer or air heater depends on the sulphur content of the coal fired, and there is a lower limit of flue gas temperature that can be attained. Below this limit the dew point is reached, resulting in costly corrosion. In short, the sulphur in the coal will either cost money for replacing equipment or some efficiency of the boiler unit must be sacrificed. The latter is usually chosen as less expensive.

Ash Content: The quantity of ash in the coal decreases proportionately the amount of energy-producing material in each ton of coal. Any increase in ash content requires the handling, removal, and disposal of large quantities of the material, resulting in increased operating and maintenance expense. The efficiency of the boiler unit is usually reduced when burning coals of high ash content because of the increased amount of combustible entrained with the ash and carried into the refuse. When high ash coal is burned in the pulverized form, the ash is always detrimental because more of it must be removed from the furnace and boiler settings, and unless ash collection is designed for the higher ash content, increased stack discharge is a nuisance for the people living in the surrounding area. Some ash is necessary to protect the grates in underfeed stoker firing, but when burning high ash coal, a larger percentage of the fuel bed consists of ash that tends to retard combustion. The quantity of ash as well as its composition has a decided effect on plant operating expense because of the increased maintenance required on the ash handling, coal handling, preparation and burning equipment.

Swelling Index: In the large stoker-fired plant, the coking property of the coal has a decided effect on operating control of fuel bed conditions. Many tests have been formulated for determining this coal characteristic but the subject is complex due to the widely varying composition of the coals which has a marked effect on coal coking quality. The amount of swelling of the coal when heated has been correlated with the coking properties so that it indicates somewhat the coking quality of the coal. The range of values of this index varies from one to nine in half numbers with coking properties increasing with the index number. Fig. 1 shows some of the coke buttons obtained by this test, which covered a range of values, and shows how the swelling varies for different coals. This index is most important to The Detroit Edison Co. in mixing coals with about the same swelling number for shipment and burning at the stoker-fired plants. Where it is necessary to burn a mixture of coals from many mines and seams, better control of the fuel bed and more stable fires can be obtained, usually resulting in higher efficiency if coals with about the same coking characteristics are mixed for burning. Since it usually is not possible to mix coals thoroughly, air distribution through the fuel bed is more difficult to control when coals with different coking characteristics are burned on the stoker at the same time. Because of unequal combustion air distribution, coal burning rates on some parts of the stoker are exceedingly high with resultant high temperatures causing burned iron and high stoker maintenance. The uncoked particles of the free burning coal may be blown either into the ash pit with high carbon loss or carried into the gas stream where the ash may collect on the tubes because the burning carbon keeps the ash plastic and provides a reducing atmosphere with lower ash-fushion temperatures. In selecting and mixing coals for the plants, the coals with the highest index numbers are delivered to the plants with the highest coal burning rates, since a high quality coal with good coking characteristics is needed for high stoker ratings.

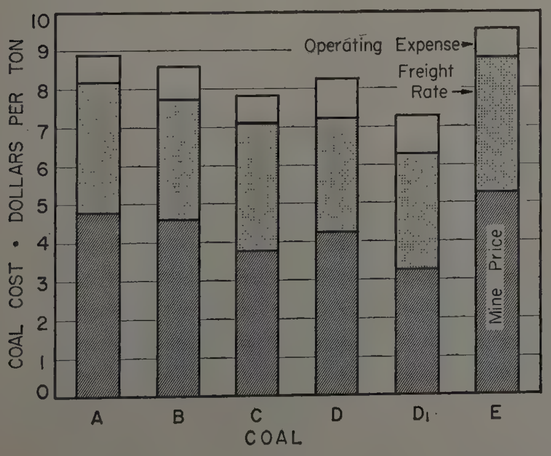
Volatile Matter: Depending on the completeness of its combustion, the volatile matter of the coal may or may not result in some decrease in steam generation efficiency. About 30 pct of the heat in the coal is contained in the volatile matter because of the high heating value of the constituents; a small amount that might escape without being burned can affect the efficiency considerably. In the selection of coals, the bituminous coals may be classified broadly as high and low volatile coals, and it is not usually practical in most furnace designs to burn these coals interchangeably because furnace design changes and changes in operating control are required. Since a greater amount of gas is given off in the pulverized fired furnace, the high volatile coals ignite and burn more rapidly than the low volatile coals and produce a richer gas and air mixture in a shorter time. In order to approach this condition with low volatile coals, it is necessary to reduce the primary air so that a richer mixture can be obtained at the burner. It may be necessary, however, to reduce the air flow to a point where the air will not pick up the coal in the mill, and the quantity and temperature may be insufficient for providing adequate evaporation of moisture in the mill to prevent delayed ignition and unstable fires. By increasing the secondary air pressure, greater turbulence is created, resulting in more rapid ignition. Besides these changes in control, it may also be necessary to provide some reradiating refractory surface in the burner zone to bring the coal up to ignition temperatures more rapidly.

Grindability: The grindability index is of decided value not only in selecting coals for existing plant mill capacity, but also in installing new mill capacity where consideration should be given to the range of grindability of the coals that may be purchased and burned. Coals with low grindability may reduce mill capacity to the point where the coal prepared and fed to the furnace is not sufficient to generate the required amount of steam for plant load requirements. Grindability is reported as an index number, and the higher the index number, the easier it is to grind the coal. Although mill characteristics vary considerably, for some coal mills, where all other components such as moisture content, size of feed, and fineness of product are the same, the capacity of the mill increases almost directly with the grindability of the coal, so that when grinding coal with a grindability index number of 100, the mill capacity is about double that obtained when using 50 grindability coal. The energy input to the mill is the same regardless of grindability, so that when mill capacity is increased, the energy per ton is decreased accordingly. Usually the low volatile bituminous coals are of softer structure and easier to grind, and a finer product is obtained from the mill. With a decrease in volatile matter, the coal must be ground finer if rapid and complete combustion is to be obtained. Unless the coal fineness is increased, the carbon loss in the refuse increases with a resultant efficiency loss. When changing to a harder coal, the size of product from the mill should be checked to determine if coarse particles are being fed to the furnace. The larger particle size not only increases carbon loss but also tends to aggravate slag collection on the tubes because of the sustained heating of the ash associated with the combustible matter and because of the reducing action of its combustible matter on the ash constituents.

Of course, the actual suitability of any coal for use by The Detroit Edison Co. depends on its performance when burned in the plant equipment. In any case where the analysis indicates that there might be a question of the coal's causing operating difficulties a burning test is conducted and the coal judged on the merits of its actual performance.

The trend of some of these coal characteristics during the past ten years is shown in Fig. 2. The average ash content increased from a 1940 average value of about 7.0 pct to a maximum of 12.0 pct, and the average heating value decreased from about 13,500 Btu to 12,300 Btu. If the reduced heating value, together with increased coal costs and freight rates, is considered, coal costs have more than doubled during this period. During the major portion of this period, it became a question of obtaining coal for company needs rather than selecting coal on the basis of quality or suitability for the burning equipment. With coals unsuitable for the equipment and high boiler ratings, many operating difficulties were experienced together with increased plant operating and maintenance expense. Fig. 2 indicates a gradual improvement in coal quality during the past two years. It is questionable whether these values will return to previous low levels. With the increase in coal tonnage mechanically mined and cleaned by washers, some increase in moisture content can be expected, and there is an economic limit to the amount of impurities that can be removed without an excessive loss of valuable coal.

Fig. 5—Comparative cost of coal, competitive coal fields.



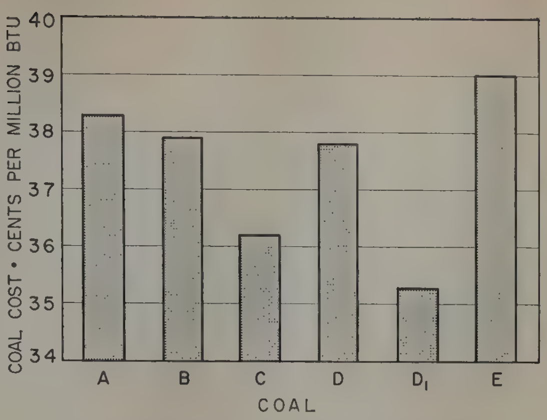


Fig. 6—Comparative cost of coal, competitive coal fields. Based on heat absorbed in steam generating unit.

As in practically all of the electric utility industry, a rapid increase in load growth and output has been experienced by The Detroit Edison Co. during the past 10-year period with a resultant increase in coal consumption. The trend for this period is shown in Fig. 3 and indicates an increase in power plant coal consumption of about 132 pct based on consumption in 1940. Since all indications point to a continuation of this growth, the coal industry will be called upon to meet ever-increasing coal requirements.

In conjunction with increased coal consumption is the need for increased handling and storage facilities. As mentioned previously, the aim of the utility is service, and a protective coal storage supply must be provided so that continuity of service can be maintained during emergency periods. The tonnage of coal in storage at this company's power plants for the past seven years is shown in Fig. 4, which reveals the emergency condition encountered and the unsettled pattern for fuel procurement and company protective coal storage. Anticipating a greatly increased demand for coal occasioned by the war, protective coal stocks were built to abnormally high levels during the early war years and burned later when coal was in short supply. Twice during this period it was necessary to install oil-burning equipment and burn oil in rather large quantities to conserve coal stocks. From an economic viewpoint, the burning of oil could not be justified since the equiva-

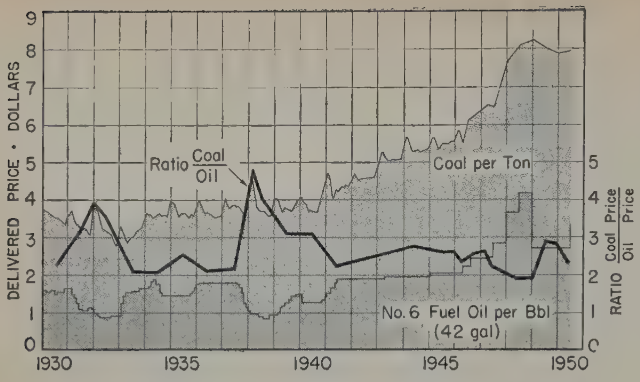


Fig. 7—Comparative prices, coal and No. 6 fuel oil.

lent oil price was about twice the coal price and resulted in higher operating fuel costs.

Evaluation of Coal from Competitive Fields

The evaluation of coals from competitive fields must be considered when building new or rebuilding old plant equipment. Trends of supply and price must be studied in an attempt to foresee what coals may be economically purchased and burned in this equipment. An evaluation of the coals must be made in order to eliminate coals that from a competitive standpoint probably will never be purchased for the new or rebuilt plant equipment and would not justify added plant investment to handle them. As shown in Fig. 5, comparative coal costs for some of the competitive coals have been plotted. To obtain the heat absorbed in the steam generating equipment when burning different coals, it is necessary to consider the coal heating values and steam generation efficiencies. With this information the cost per million Btu absorbed in the steam generator may be calculated, see Fig. 6. Although many comparisons are made on the basis of delivered coal prices alone, the competitive relationship can be changed completely, depending on the coal heating values, steam generating efficiencies and plant operating expense. The names of the coal fields have been purposely omitted since the coal analysis is representative of only certain districts of these fields. The evaluation is based on the burning of these coals in a pulverized fuel fired furnace of liberal design. It would not necessarily hold for other types of firing or when attempting to burn coals of widely varying properties in equipment already installed. The properties of these coals from the various fields vary widely, with the ash-softening temperature ranging from 2100°F to 2600°F. A design to cover this range would not be justified unless the coals are competitive, and the savings that may be effected can justify the additional investment. From an operating standpoint, furnace design should be liberal in respect to heat transfer rates so that continual operating troubles, decreased availability and increased operating expense will not be encountered during the life of the equipment.

Besides making an economic study and evaluation of coals that might be competitive, it is also necessary to consider the dependability of supply and the quality of product that may be expected in the future. Change in coal quality may increase plant operating and maintenance expense materially besides decreasing the availability of the equipment. Of course, any change in the relationship of mine prices, freight rates or quality of coal between the coal fields may change the picture completely. As a basis for forecasting the trends of mine prices, the published mine production costs for the various districts is useful. Although freight rates have increased considerably in the past few years, the relationship of transportation costs between the various fields has been established over a long period, and it is doubtful that any radical change will occur in the near future.

Evaluation of Competitive Fuels

Although the delivered price of coal from various fields usually bears some relationship, the price of competitive fuels such as heavy fuel oils or natural gas may fluctuate widely and not change necessarily in relation to coal prices. The geographic location of the utility in relation to fuel source and refining areas, together with relatively cheap water-borne transportation freight rates, determines to a great extent the possibility of heavy fuel oil being competitive with coal. Where the fuel cost is only a small part of the total production cost of the product and the characteristics of this fuel are desirable in the manufacturing process, industry is willing to pay a premium to obtain this fuel. When industrial demand was strong, prices remained firm and only competed with coal during periods of business inactivity when heavy oil came on the market at distress oil prices. In Fig. 7 the price of heavy fuel oil and coal has been plotted for a 20-year period. Examination of the price ratio indicates that there were only two periods when heavy oil was competitive with coal for The Detroit Edison Co. use as fuel. These are published prices and some oil was sold at lower distress prices. The company's competitive position in relation to this fuel is not similar to the coastal areas where large quantities of foreign heavy oil imports have been received and burned during the past few years.

Although oil may not be competitive with coal, there may be times when the utility is justified in paying a premium to burn this fuel in limited quantities to increase steam generation, coal preparation, or handling capacity for peak demands. The additional fuel expense for peaking operation may be more than justified when considering the investment required to install additional equipment that might be needed for only short periods during the year. At two of the company's plants, oil burning facilities have been installed, and it is expected that this fuel will be burned in limited quantities when needed for peak load demands.

Although the industrial market price for natural gas is not normally competitive with coal, rates for off-season supply might bring natural gas into the picture at various times. When gas is supplied on this basis, the contract usually contains a provision that the gas may be cut off after a stipulated notification period, and it is necessary to maintain a firm coal supply with a resultant decrease in the amount of saving in operating expense that can be credited to the gas.

With increased oil imports, extension of natural gas lines into new areas, greatly increased coal productivity due to mechanization, the competition between various fuels has become keener. The amount of regulation these various fuel industries experience as to price and end use of these fuels will affect their competitive relationship, and it is just as difficult to forecast fuel trends as it is other business trends under present conditions.

Coal Preparation in England And Holland

by John Griffen

Methods of coal preparation in England, including usage of American units such as the Chance sand flotation process and Denver flotation cells, are compared with methods used in the United States. Problems of the coal industry in Holland are summarized.

OF the western European countries, only England has made any extensive use of equipment developed initially by the coal preparation industry of the United States. About 20 years ago, the Chance sand flotation process for cleaning coal was introduced in England, and it has had a fairly wide adoption, particularly in the South Wales coal fields. In recent years, particularly since 1946, a considerable number of Denver flotation cells have been installed throughout the English coal fields for the cleaning of slurry. Until the advent of the Denver cells, the flotation of coal slurries in Western Europe had been done largely with the Mineral Separations cell or its modifications. Kleinbentik of Dutch State Mines had developed a cylindrical cell used in Holland and occasionally in Belgium and France.

In England, the establishment of the National Coal Board to control the planning and operation of all coal production has retarded somewhat modernization of properties. A considerable period of study was required before definite long-range plans could be formulated. It was soon recognized that stopgap measures were needed in many cases to bridge the period until final plans could be developed and consummated.

One of the stopgap problems in coal preparation was providing plants of small and medium capacity that could be used for a few years at one site and yet could be moved economically to another site so as to reduce depreciation charges per year and per ton. Such plants are being installed but the most satisfactory answer probably has not yet been found. One such installation uses a Norton Baum jig to wash 6 in. to 0. The equipment is in a steel structure rather than in a reinforced concrete frame with brick curtain walls, which is the normal structure for permanent plants.

Since England has between 700 and 800 preparation plants for an output of 200 million tons per year, one of the long-range objectives is a consolidation of mine output which will require fewer coal

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preparation plants since each will have a larger capacity than the usual plant. The ideal plant capacity had not yet been fixed but, for the present, cleaning plants of about 800 tons per hr seem to be favored. Actually, these are usually two plants of 400 tons per hr input in one structure. This duality is often dictated by the fact that several seams are mined producing coals of different qualities that must be prepared separately in the sizes above slack for different markets.

The four phases of coal preparation in which England has made further progress than the United States are:

1. Mechanical cleaning of large coal to eliminate picking labor.

2. The use of heavy media processes to clean difficult coals and particularly to produce two coal products, a very high quality first grade coal for special purposes such as railway fuel, London house coal and the export market, and a medium quality coal for ordinary uses.

3. The cleaning of slurries by froth flotation.

4. The use of starch and other coagulants on the feed to thickeners so as to completely clarify washery waste water before re-use and to prevent stream pollution.

There are a number of plants regularly cleaning 8 in. to 2 in. raw coal, thus reducing the number of pickers required to clean the lump. At one plant, the +8 in. lump passes over a picking table where about three pickers remove rock larger than about 8 in.; the remainder passes under a pick breaker that breaks coal and bone to about —12 in., although coal lumps larger than 12 in. enter the bath. The large coal then joins the 8 in. to 2 in. size and is cleaned in heavy medium baths. In the first bath, coal is separated at about 1.47 sp gr and final refuse at about 1.70 sp gr. The middlings are crushed —2 in. and added to the raw 2 in. to 0 for subsequent cleaning. The coal product of the first bath is recleaned in a second two-product bath at 1.33 to 1.37 sp gr making two qualities of coal:

First grade, +4 in., 5.0 pct ash;
First grade, 4x2 in., 4.5 pct ash;
Second grade, +2 in., 15.0 pct ash.

This coal has rather high inherent ash content.

Heavy medium plants treating coals of lower inherent ash content are producing washed coal as follows: 5 in. to 3 in. round, 1.7 to 2.2 pct ash; 3 in. to 2 in. round, 1.9 to 2.8 pct ash; 2 in. to 1 in. round, 2.2 to 2.9 pct ash; 1 in. to ½ in. square, 2.7 to 3.2 pct ash.

In the cleaning of slurry, not only is froth flotation widely used, but the slurries are generally coarser than in this country. Slurry —½ mm or 28-mesh is usual, and with the bituminous coals of South Wales and the gas coals of Durham, which are quite easily floated, material as coarse as 1/20 in. or nearly —10-mesh Tyler standard screen scale is handled. The South Wales float coal is as low as 3.0 pct ash.

Although closed water systems and the use of coagulants are not by any means universal, the larger, newer plants, even those built just before or during World War II, were generally so operated, and in every case the thickener overflow was crystal clear.

The English coal market has been accustomed to nut and larger sizes of extremely low ash content, 4 pct or less, which were readily produced from the excellent seams of the past. The exhaustion of those seams and the changes in mining methods are resulting in run-of-mine output much more difficult to prepare, but all efforts are directed toward the building of preparation plants that can continue to meet these rigid standards of high quality products in these larger sizes. The requirements for slack and nut-slack are not so rigid, and in most cases the cleaning of these sizes is done by methods used in the United States.

The coal industry of Holland, in marked contrast to that of England and the United States, is on a small scale, and there are very few coal producers. In 1938 the Dutch coal industry produced 13½ million metric tons. In plants operated by the industry, part of this coal was processed into 2.4 million metric tons of coke and 1.2 million metric tons of briquets. As the result of the war, coal output dropped to 5 million metric tons in 1945 but by 1948 had increased to 11 million metric tons. The changes in production of coke and briquets roughly paralleled that of coal.

The government-controlled Dutch State Mines, which operates four mines, produces 60 pct of the output. Four private companies, which operate eight mines, are under a certain amount of government control.

The four Dutch State Mines properties with an annual output of about 8 million metric tons are large properties, and, as the seams are generally deep, up to 1000 m of cover, mining is difficult and expensive and the run-of-mine coal is quite dirty, containing 30 to 35 pct reject.

Because of limited reserves, the high cost of coal and its importance to the economy of Holland, the Dutch State Mines are carrying on extensive research on mining and preparation problems and have large laboratory facilities and research personnel.

In preparation, the Dutch State Mines has lead in the development of heavy medium processes—the Barvoys, Loess, Driessen—and now the Dutch State Mines process using flotation tailings or ground washery refuse as medium solids. Another excellent heavy medium process, the Tromp, was developed at Domaniale, one of the private mines. The Tromp process utilizes fine magnetite, mill scale, or other heavy solids, which may or may not be magnetic, and recovers and reconditions the medium solids by gravity rather than magnetic means.

Much of the effort of the Dutch State Mines in developing heavy medium processes has been directed by the desire to use medium solids readily available in Holland. The barytes required for the Barvoys process must be imported from Germany. Magnetite has to be imported from Sweden. This probably explains the interest in loess, which is plentiful in Holland. However, its technical limitations are focusing attention on flotation tailings and ground washery refuse as medium solids.

Since coal in Holland is so valuable, preparation plants recover high ash middlings, and the refuse rejected is almost pure rock, usually analyzing 75 to 85 pct ash. In addition to producing its own electric power, the Dutch State Mines operates a large generating station which supplies electricity to the National Electric Grid.

Practical Design Considerations for High Tension Belt Conveyor Installations

by J. W. Snavely

THE high tension belt conveyor is introducing a new and tremendously expanded era of low cost bulk material handling. High tension belt conveyors are generally those installations involving very long centers, high lifts, or drops, in which the belts are stressed up to their maximum tension values, and further, where the belt construction provides tension capacity far beyond what is possible with conventional belt constructions. With these high tension installations, the magnitude of the forces involved demands careful refinement of accepted design practice in order to achieve optimum balance of all factors.

No attempt will be made to evaluate the relative merits of belt conveyor haulage with other means of transportation. For present purposes, it is assumed this has already been done in favor of belt conveyor. Neither will any attempt be made to evaluate the various conveyor belt constructions now available or to balance the advantages of various types of mechanical equipment.

It is also assumed that the basic haulage information on which the conveyor design is based is accurate and complete. A sustained maximum, uniform load on the belt at all times must be achieved through proper feed control and the use of adequate surge storage to level the peaks and valleys of any varying demand for the material being handled.

General Belt Capacity Considerations

The belt conveyor capacity tables published by various belting and conveyor equipment manufacturers vary to a considerable degree, and the ratings given are quite conservative. Of necessity, these published ratings are based on the handling of average materials under average conditions.

In applying a high tension belt, all possible capacity from the belt must be obtained in order to hold its width to a minimum and thereby limit the initial cost. Two factors are involved, loading to maximum cross section area and traveling at a maximum practical speed.

Belt Loading: Proper treatment of the loading of the belt will result in maximum cross section to the load, and published capacity ratings can be exceeded, sometimes by appreciable margins. On the 10-mile conveyor haul used in the construction of Shasta Dam, California, although the rated capacity of the belt line was 1100 tons per hr, at times the system handled peak loads of 1400 tons per hr, almost 25 pct better than the rated capacity. One of the large coal companies has been able to exceed rated capacity by as much as 50 pct.

Loading conditions which must be controlled are:

1. Large lumps must be scalped off and rejected or the load must be primary crushed before being placed on the belt.
2. The material weight per cubic foot must be accurate, must be known for all the materials being handled, and must be known for the complete range of conditions of the individual material being handled. Long centers and high lifts magnify small differences into serious proportions.
3. Uniform feeding to the belt is most important. Various types of feeders are available, which can be used to place a constant predetermined volume of material on the belt, or, where an appreciable range of material weight exists, through electrical control actuated by current demand, to place a predetermined uniform tonnage on the belt. One long slope belt in a coal mine in Pennsylvania is being fed at three separate stations with the controls so arranged that whenever the maximum load is going onto the belt from the first station, the other two stations automatically cut out. Whenever the load from the first station drops back, the other two stations again automatically cut in.
4. Careful design of the chutes and skirts is necessary to get the load centered on the belt with a minimum of free margin along each edge. Some free margin at the edge of the belt is necessary to prevent spillage, but if the load can be kept accurately centered, this free margin area can be reduced, and more material can be carried on the belt. What can be accomplished in this respect will vary widely, depending on the nature of the material being hauled. The chute and skirt design must also protect the belt.
5. The design of chutes and skirts should also get the load traveling in the same direction and close to belt speed, so that the load comes to rest on the belt as quickly as possible. The design of the chutes and skirts is worthy of careful study, and after a system is put into operation it should be experimented with to get the best results.

Belt Speed: High belt speeds should be used in high tension work. Obviously, high belt speeds enable haulage on a narrower belt, reducing initial cost.

The major portion of belt wear takes place at the loading point and around the terminal pulleys. The

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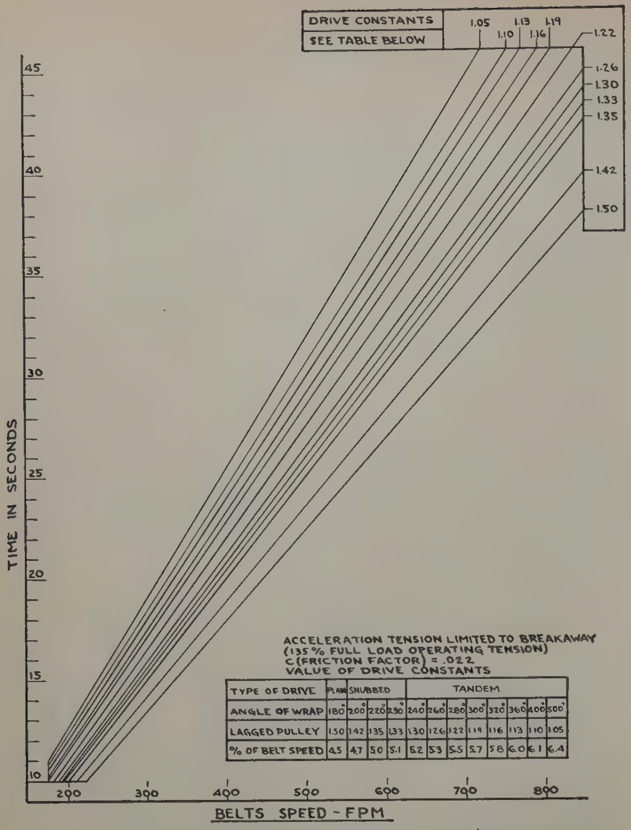


ACCELERATION TENSION LIMITED TO BREAKAWAY
(135 % FULL LOAD OPERATING TENSION)
C (FRICTION FACTOR) = .022
VALUE OF DRIVE CONSTANTS

TYPE OF DRIVE	PLAIN	SNUBBED			TANDEM							
ANGLE OF WRAP	180°	200°	220°	230°	240°	260°	280°	300°	320°	360°	400°	500°
LAGGED PULLEY	1.50	1.42	1.35	1.33	1.30	1.26	1.22	1.19	1.16	1.13	1.10	1.05
% OF BELT SPEED	4.5	4.7	5.0	5.1	5.2	5.3	5.5	5.7	5.8	6.0	6.1	6.4

Fig. 1—Minimum acceleration time for high tension belts for horizontal and inclined conveyors.

wear that takes place at the loading point is primarily cover wear, resulting from the scuffing action of the material pouring onto the belt. In addition, there may be damage to the carcass from the impact of large lumps. The wear that takes place at the terminals is primarily to the carcass, from the effect of the flexing around the pulleys. Both types of wear become sharply reduced as the time cycle of the belt lengthens.

The time cycle of a belt can be defined either as the length of time required for any given point of the belt to travel completely around the conveyor, or the number of times per unit of time a given spot on the belt passes under the loading point. Dividing the total length of the belt by the belt speed, or roughly, twice the conveyor centers by the belt speed, gives the time cycle. For example, a conveyor of 1000-ft centers and 500-fpm belt speed would have a time cycle of approximately 4 min.

Long center units permit high belt speeds without seriously shortening the time cycle, and flex life as a limiting factor on belt life can be greatly discounted. Cover wear then determines belt life, and if a given point on the belt passes under the loading point but once every 5 or 6 min, or longer, the life of the cover can be measured in terms of millions of tons handled.

At high belt speeds, the design of transfer becomes most important, both from the standpoint of maintenance cost and material degradation. The degradation of coal at a transfer can be a serious and costly problem. However, in the design of transfers much can be done to minimize the amount of degradation and to make high belt speeds entirely practical.

All of the component parts of the high tension belt conveyor require particular study, the belt, the idlers, the terminal machinery, the drive, and the framework. Even though each of these components must be individually treated, the problems arising from the high belt tensions can be solved only through interdependent component designs. The most important component, because it represents the largest single item of cost, is the conveyor belt.

Selecting the High Tension Belt

The selection of any conveyor belt results from proper balance of strength and quality factors. The high tension belt requires a far more thorough analysis of the strength factors and their determinations than has been necessary for the conventional installation. But as to the quality factors, mainly the covers and rubber used, the application of the high tension belt does not involve any new considerations other than possibly the effect of the loading point time cycle on cover thickness.

Complete analysis of the strength factors involves the calculation and analysis of the belt tensions, with particular emphasis upon the evaluation of horsepower determination formulas, the treatment of slope, and acceleration tensions. The steps involved in determining the tensions for the high tension belt application are the same as for any conventional belt except that all steps must be completely analyzed, particularly for the effect on the total tension. These steps are essentially the following: (1) Horsepower determination, (2) maximum belt tension determination including: (a) effective or working tension, (b) slack side or initial or drive friction tension, (c) slope or weight of belt tension, (d) breakaway or starting tension, (e) acceleration tension.

Horsepower Determination: In the published literature on belt conveyor design, there is agreement in the calculating of the various tensions after the horsepower has been determined, but there is a confusing variety of methods of horsepower determination, with equally confusing results. Inasmuch as high tension applications use substantial amounts of power, it is highly important that the horsepower determination be accurate, and a study of this factor is particularly pertinent.

The power required at the headshaft of any belt conveyor is the algebraic sum of the following four factors: (1) Power to run empty conveyor, (2) power to move load horizontally, (3) power to elevate load, or power generated in lowering load, (4)power to operate trippers.

Only the first and second factors are discussed here. The power needed to lift or lower the load contains no controversial elements in its determination, and the fourth factor will seldom, if ever, be great enough to have differences in its calculation affect the final figure except very slightly. The first two factors evaluate the friction losses of the conveyor with considerable variation of treatment. These differences largely center around two things, the friction factor used to measure the power loss in turning the idler rolls and the measuring of the power loss in the terminal machinery.

Idler Friction Factor: High tension belt applications involve the use of only the highest grade antifriction bearing belt idler equipment. Even with uniformly high-grade equipment, there exists a lack of uniformity in the idler friction factor recommended by different authorities. There is a rather wide variation, some of which is accounted for by the fact that the application of the friction factor is influenced by several considerations not subject to precise determination. Among such influences are the maintenance of the conveyor including the care of the belt, care of the idlers, cleaning of spillage, the greasing and type of lubricant used, as well as the construction and alignment of the conveyor frame, exposure to the weather, variations in load, spacing of the idlers and diameter of idler rolls. Aware of the effect of these unknowns, from experience the conveyor equipment manufacturers have generally used the conservative 0.03 friction factor. For the average industrial application, the use of this factor has worked out successfully over a long period of time.

Comprehensive studies of friction losses, notably on the 10-mile Shasta line and on one of the earlier long distance coal handling installations, have demonstrated that the 0.03 factor is unnecessarily conservative when all elements of the conveyor design and construction are well carried out. The scatterband of the readings from Shasta showed that 0.022 was never exceeded.

For the extremely long center belt conveyor it is possible to refine further the use of this friction factor and recognize one more condition which the studies revealed, namely, that the friction losses in moving an empty belt are less than when moving the same belt when loaded. The load produces additional sag of the belt between idlers, and some power is required to lift this load the slight amount over each idler. Further, there is consumption of power by the load in coming to rest on the belt after being discharged onto it and from subsequent changes in the load cross section. The studies made by Hetzel and Albright established a friction factor of 0.015 for moving an empty belt, which is the one used in the following belt conveyor horsepower formulas for maximum tension installations:

(Antifriction bearing equipment exclusive of tripper requirement. Horsepower for tripper equals power consumed by tonnage lift through tripper plus power to propel. See manufacturers' catalogs.)

For average quality installations, industrial and field conveyors,

$C = 0.03 \qquad L_o = 150$

For high quality installations, conveyor transportation lines,

$C = 0.022 \qquad L_o = 200$

$$\overset{\text{(At Headshaft)}}{hp} = \overset{\text{(To Move Empty Belt)}}{\frac{C(0.03QS)(L+L_o)}{990}} + \overset{\text{(To Move Load Horizontally)}}{\frac{CT(L+L_o)}{990}} \pm \overset{\text{(Lift or Drop)}}{\frac{TH}{990}} \qquad [1]$$

For maximum tension installations, slope belt conveyors,

$C = 0.015 \quad C' = 0.022 \quad e = 0.007 \quad h = 0.004$

$$\overset{\text{(At Headshaft)}}{hp} = \overset{\text{(To Move Empty Belt)}}{\left[\frac{C(0.03QS)L}{990} + e(0.03QS)\right]} + \overset{\text{(To Move Load Horizontally)}}{\left[\frac{C'TL}{990} + hT\right]} \pm \overset{\text{(Lift or Drop)}}{\frac{TH}{990}} \qquad [2]$$

where C = idler friction factor
C' = idler friction factor
Q = weight of moving parts, lb

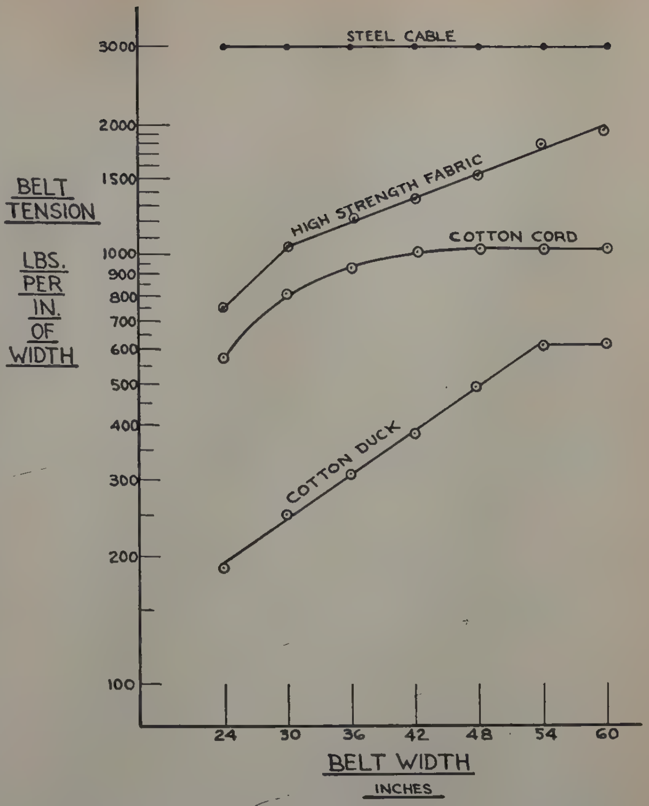


Fig. 2—Conveyor belt high tension rating curves by belt widths and construction.

S = speed of belt, ft per min
L = horizontal centers, ft
L_o = centers correction, ft
T = tons per hr, 2000 lb
H = lift or drop, ft
e = correction factor for added hp, empty belt
h = correction factor for added hp, moving load horizontally

Centers Correction Factor: The power consumed by the belt in going around the terminal pulleys must also be considered. This factor is largely independent of the conveyor centers and its early treatment, while satisfactory for the average length conveyor, overpowered the long center conveyor. Many handbooks of 20 years ago determined terminal friction losses by arbitrarily increasing the power for moving the empty belt and for horizontal conveying by 20 pct for conveyors up to 50 ft in length, 15 pct for 50 to 100 ft, 10 pct for 100 to 150 ft, and 5 pct for conveyors over 150 ft centers.

A more accurate and convenient method was worked out by Goodyear and consists of adding a length constant to the actual horizontal conveyor centers. Graphical analysis of a number of studies showed that with antifriction bearing idler equipment, the terminal consumption of power was about equal to an arbitrary increase in horizontal centers of 200 ft. This 200-ft length constant, or L_o constant by which it is usually designated, is added to the horizontal conveyor centers when calculating the horsepower, as shown in formula 1. The calculated horsepowers using this method are reasonably close to observed operating power demand. In some cases other values have been used for L_o, but considering current experience, this would seem to be unnecessary.

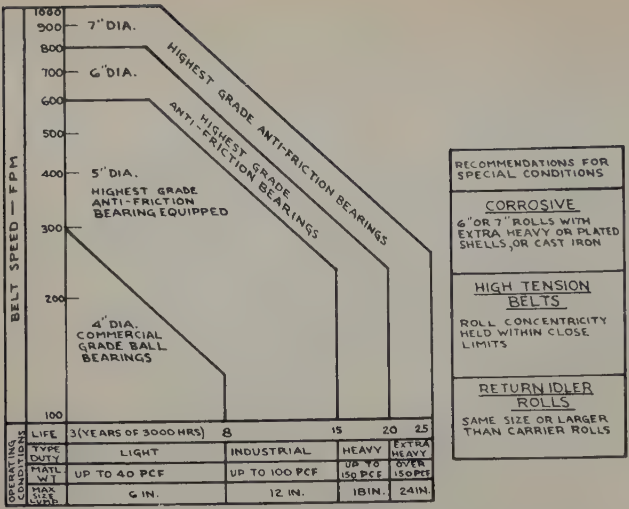


Fig. 3—Selection chart for belt carrier idlers with steel rolls.

Above curves are based on shell life, and correct roll size is shown in area in which it falls. Select correct operating conditions. Use roll diameter shown in this area if belt speed falls within range for chosen area of operating conditions. If belt speed is faster, use roll diameter in area opposite belt speed. If belt speed is slower, use roll diameter in area above operating conditions.

Where maximum stressing of the belt is being applied, it again becomes desirable to refine further the calculation of the terminal factor to provide maximum accuracy. Hetzel and Albright have worked out individual factors to be applied respectively to the empty belt and horizontal conveying formulas, which are reflected in formula 2.

Quality of Installation: It is apparent that difference in treatment exists, but rather than being contradictory, it should be observed that the degree of conservatism used depends on the type of installation involved. Table I shows the different results obtained in calculating the horsepower for the same problem using different current equipment and belting manufacturers' handbooks, together with formulas 1 and 2. At face value, these varying results are confusing, but a reasonable pattern emerges when the logic of the foregoing discussion is applied, and a tabulation of the type of installation for which the formulas were basically intended is superimposed on the results. The thoroughness with which the high tension belt conveyor is engineered eliminates most of the indeterminate factors in industrial and field conveyor installations, and sound engineering consequently dictates the use of the most advanced formula for maximum accuracy and balance of design.

Table I. Comparison of Belt Conveyor Horsepower Requirement

Calculated from Formulas 1 and 2 and from Leading Manufacturers' Catalogs

Problem: Required hp at headshaft to convey 1000 tons per hr of coal weighing 50 lb per cu ft, up 15° incline at speed of 660 fpm, 1200 ft inclined centers (1160 ft horizontal centers) and 311-ft lift.

Kind of Installation	Hp to Move Empty Belt	Hp to Move Load Horizontal	Hp to to Lift Load	Total Hp at Headshaft
Avg Quality Industrial and Field Conveyors				
Formulas 1 and 2	39.3	39.7	314.1	393.1
Manufacturer A	42.0	41.2	314.1	397.3
B	40.5	39.7	314.1	394.3
C	40.0	38.7	314.1	392.8
High Quality Conveyor Transportation Lines				
Formulas 1 and 2	29.9	30.2	314.1	374.2
Manufacturer D	33.2	37.4	314.1	384.7
E	30.3	30.2	314.1	374.6
Maximum Tension Slope Belt Conveyors				
Formulas 1 and 2	24.3	29.7	314.1	368.1
Manufacturer F	25.5	31.3	314.1	370.9
G	25.8	25.8	314.1	365.7
H	20.9	24.9	314.1	359.9

Beyond the treatment of the friction factors, there is little need for further discussion of these formulas, for there are no other differences. It must be emphasized again that the peculiar requirement of the high tension belt is a matter of degree only. The magnitude of the forces involved seriously enlarges what are normally insignificant differences and every possible refinement is necessary to a final balanced design.

Maximum Tension Determination: Having accurately determined the horsepower, the maximum tension for the belt must be established. This is nothing more than a matter of arithmetic, but because of the interrelation of the equipment components, trial computation is required before the best combination can be found. The formulas for determining the slack side or drive friction tension and the other respective tension elements that build up the maximum tension are given in any number of handbooks.

The high tension application requires full consideration of several of the lesser tension demands which, for the ordinary conveyor, have minor importance and can be often ignored. The belt tension demands that must be studied are the slope, acceleration, and breakaway tensions.

Slope Tension: The slope tension found on all inclined or declined units is the weight of the belt hanging from the head pulley, minus the friction of the idler equipment. With slope belts of long inclined centers, the weight of the belt hanging from the head pulley imposes a considerable amount of tension, and it is a factor that must be considered. At the same time, this slope tension must be put to work for it frequently will supply all or most of the necessary slack side or drive friction tension, even for large horsepowers. When it is possible to locate a horizontal gravity type of takeup at the tail end of a slope conveyor, this means that the weight of the belt is supplying practically all of the counterweight tension that otherwise would have to be added.

Acceleration and Breakaway Tensions: Because the high tension belt is highly stressed, the additional tension introduced during the accelerating period when it is started must also be taken into consideration. By predetermining the acceleration time, the additional tension introduced during the starting period can be held to a reasonable figure. With proper electrical control equipment, the acceleration period can be readily set at any desired length of time, 20, 30, or up to 60 or more sec.

The design of the starting controls should provide long enough acceleration time so that the accelera-

Table II. Typical High Tension Ratings, Cotton Cord Belts, Lb per in. of Belt Width

	GOODRICH			GOODYEAR		MANHATTAN	
No. Plies	No. 50	No. 70	No. 100	Compass No.		Ray-Man No.	
4	206			30	200		
5	258	350	457	40	250	33	480
6	309	432	573	50	300	34	600
7	361	512	688			35	720
8	412	592	804	100	450	46	840
9	464	672	918			47	960
10	515	752	1032	125	600	48	1100

tion tension is held at approximately the breakaway tension, that is, the tension required to start the belt and its load from rest and get it moving. Breakaway tension is approximately 130 to 140 pct of the full load effective tension, with the usual design figure being 135 pct. In starting, just enough tension must be supplied to the belt to get it moving, after which the power to get up to full speed must be supplied slowly so that the starting tension is never exceeded. With control of the starting condition, it is not necessary to build as much reserve strength into the belt as would otherwise be necessary, or conversely, for a given strength of belt, maximum effective work can be obtained from it.

It is important therefore that the magnitude of the acceleration forces be accurately determined. It can be shown that the acceleration tension in a belt for horizontal and inclined conveyors varies directly with the belt speed, and inversely with the friction factor used, the type of drive employed, and the time of acceleration. It can further be shown that the acceleration time in seconds will vary within a range of roughly 1/5 to 1/30 of the belt speed in feet per minute. When acceleration tension is held to a breakaway tension of 135 pct, the acceleration time may be obtained quickly and fairly accurately by letting acceleration time in seconds equal 5 pct of the belt speed in feet per minute.

For the operating conditions indicated, an accurate, convenient method of finding the acceleration time in seconds is provided in Fig. 1 in which the acceleration time can be read directly if the belt speed and angle of wrap of the drive are known.

This chart is only for horizontal and inclined conveyors. The problem of a declined conveyor introduces a new condition since the peak operating tension and the peak acceleration tension do not occur at the same point in the belt. A special study of a declined unit is necessary to make sure that the accelerating tension plus the operating tension at the point of maximum accelerating tension does not exceed 135 pct of the maximum operating tension at any point in the system.

Belt Specifications

With the determination of the maximum tension requirements, the type of belt can be selected. In recent years, great strides have been made by the belting manufacturers in increasing the tension capacity of conveyor belts. This development has been along several lines, the more common being the use of cotton cord constructions, high strength fabric construction, and steel cable construction.

No one construction is best for all conditions. A complete study of all the factors is necessary each time, and great variation in the final selection can be expected. For example, with a slope belt problem, although the belting cost might be very much lower if two or three flights are used, this saving in belt cost might easily be outweighed by the cost of providing the necessary machinery rooms underground, plus the cost of drifting a large enough slope to get the heavy drive machinery down to the machinery rooms. In such an instance the use of a very high tension capacity belt, such as a steel cable belt, even though high in initial cost, could easily be the best and most economical selection.

Tables II, III and IV have been compiled from published data of the indicated companies to show what can be expected today from several typical constructions of high tension belts as made by several manufacturers. It can be readily seen from these tables that the horizon for the application of belt conveyors has been pushed back a very great distance.

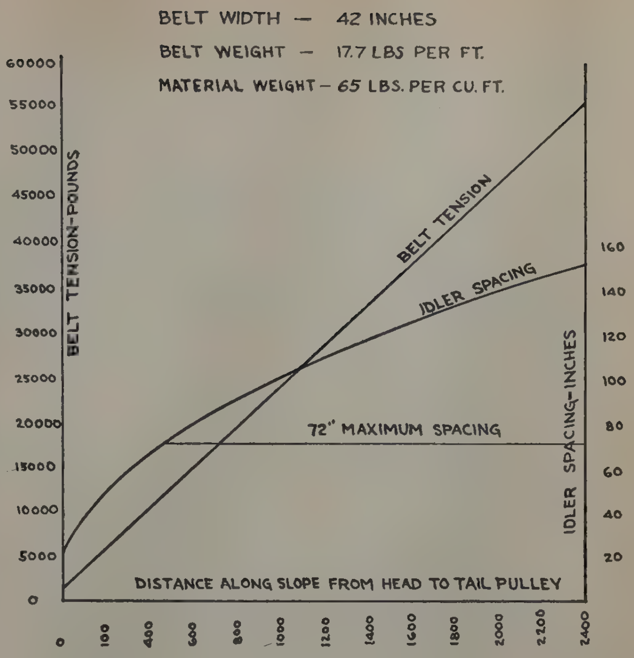


Fig. 4—Graduated idler spacing curve for high tension belts.

Hauling on the level, one conveyor in Pennsylvania has centers of 2 miles, and it could be possible to carry the 1000 tons per hr of coal in the problems of Table I, on a 42-in. steel cable belt, with 9½ miles between the loading point at the tail pulley and the discharge point at the head pulley.

Along with tension capacity, another important point in belt selection is that of troughability. A conveyor belt must be sufficiently flexible laterally so as to trough properly upon the supporting idlers. This is important both from the standpoint of obtaining the proper load-carrying capacity, and for proper training of the belt to run true both empty and loaded. At the same time, however, a belt cannot be overly flexible because it must also ade-

Table III. Typical High Tension Ratings, High Strength Fabric Belts, Lb per in. of Belt Width

		U.S. RUBBER			
No. Plies	XN	YN	85	CX	CL
4	200	280		440	600
5			425	550	750
6			510	660	900
7			595	770	1050
8			680	880	1200
9			765	990	1350
10			850	1100	1500
11			935	1210	1650
12			1010	1320	1800
13			1100	1430	1950

Table IV. Typical Maximum Tension Ratings, Steel Cable Cord Belts, Lb per in. of Belt Width

GOODYEAR		GOODRICH		MANHATTAN	
Compass No.		Steel Cord No.		Whip-wire No.	
250	900	1000	1000	17	1000
350	1200	1500	1500	19	1250
450	1500	2000	1800	21	1450
650	2500	2500	2500	26	2000
750	3000	3000	3000	37	2500

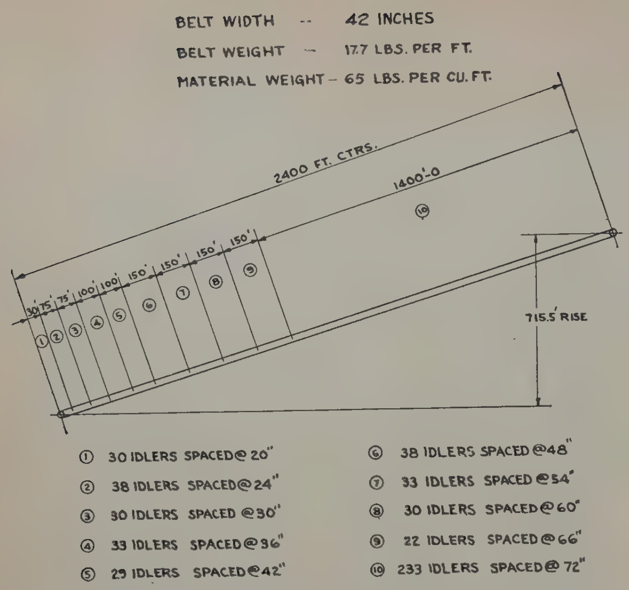


Fig. 5—Graduated idler spacing diagram for high tension belts.

quately support the load it carries; that is, there must not be excessive sag of the load, or dropping of the belt edges between idlers, or creasing of the belt in the gap between idler rolls. With high tension problems, however, because of these tensions, the ability of the belt to support its load is seldom a factor in the belt selection.

Considerable variation in the flexibility of conveyor belts results from the various constructions mentioned. The most flexible is the single plane cord construction, with flexibility decreasing as additional plies of cord or fabric are built up in forming the conveyor belt. The slope of the curves in Fig. 2 is a general indication of the troughability of the respective belt construction, with flexibility increasing as the horizontal is approached.

The lack of troughability has largely limited the tension capacity of conveyor belts in the past. While with standard fabric constructions it is theoretically possible to achieve any desired tension capacity by using the requisite number of plies, troughability has sharply limited what has been practical in the number of plies used. Moreover, it is obvious that the width of the belt also enters in as a factor, which has meant that for the popular widths of 24 to 36 in. more than seven or eight plies could seldom be used. Modern construction of belts has radically changed all that.

Fig. 2 presents the maximum ratings for the constructions shown, and for a given problem, can be helpful in indicating quickly the probable belt construction that will be required. For example, if for a 36-in. belt the maximum tension comes to 2000 lb per in. of width, it is evident that only a steel cable belt can be used. But if the tension requirement is 1000 lb per in. of width, then a high strength fabric belt can also be considered, and since that particular tension falls close to the maximum for cotton cord, this construction also should be carefully checked. Or, if the tension requirement is 300 lb per in. of width, a cotton duck construction will do the job, and the higher cost constructions of belting will not be required.

Fig. 2 is not intended and cannot be used as a basis of selection, but it can provide a quick rough check on recommendations. Assume that a proposal has been received covering a steel cable belt for a belt tension that falls within the cotton cord range. In that case the possibility of using a cotton cord construction, with possible saving in initial cost, should be checked. Ratings falling close to the curves should be particularly checked because of the overlapping of various manufacturers' maximum ratings for approximately comparable constructions.

It must be emphasized that the high tension ratings for conveyor belts that are shown cannot be used indiscriminately but can only be used when every factor of conveyor design is given highest possible quality selection. That means approval from the engineering department of the supplier on the engineering of the conveyor, on the starting controls, and the braking. Vulcanized splices are essential, and pulley sizes must be larger by at least one ply than the handbook recommendation. All of the equipment must be top quality and excellent maintenance must be provided. The belts must use top quality skim coats, and provision must be made for vulcanized repairs. Counterweighted takeups must be provided with ample provision for resplicing the belt. The installation must not be overmotored.

Belt covers and the other quality specifications must be considered. The thickness and the grade of the rubber covers, including such considerations as the tensile strength and friction of the rubber, are little different for a high tension belt than for other conveyor belt applications. The final determination of the belt specifications will depend upon best meeting the individual operating conditions, but without exception, high tension work calls for the very best quality.

One specific point to be mentioned is the effect of the time cycle on cover thickness. High tension conveyor belt applications invariably mean long centers, and this means a long time cycle. Since the major part of cover wear results from the scuffing action at the loading point, a long time cycle often makes it possible to reduce the thickness of the rubber cover. Since this is balancing cover life with carcass life, the overall service life of the belt in terms of millions of tons handled is not affected. Reducing the thickness of the belt cover means saving in the initial cost of the belt.

The long center units involved in high tension work require several splices in the belt because shipping lengths are limited to rolls of belting of approximately 1000 to 1200 ft in length or possibly slightly longer, depending on cover thickness. Only the vulcanized splice can be considered for the high tension belt since it is the only connection that permits utilization of close to the full strength of the belt. The vulcanized splice is 80 to 85 pct efficient, and equally as important, it eliminates the deterioration which is extremely difficult to stop at any mechanical splice.

The splicing of steel cable belts is a matter of considerable interest. One manufacturer uses a method of calculated dispersal of cable ends, so that the load from each discontinued cable is transferred to a group of adjacent cables through rubber in shear. Such a splice can be made to almost any desired strength depending on the extent of the dispersal of cable ends, usually approximately 80 pct of the strength of the belt itself. The cable ends are not directly joined together, but this manufacturer considers the method used, even with its slight loss of strength, as superior to attempting to join the ends of the cable together, which might result in stiff places in the cable and an unequal tension distribution from cable to cable.

One of the other conveyor belting manufacturers splices its steel cable belt by a means of joining the cable ends together, these joints occurring in a staggered pattern. A special type of connection is used to connect the cable ends, and after a preliminary fastening of the couplings, the entire spliced section of the belt is put under tension so as to obtain a uniform distribution of the tension across the width of the belt. After equalizing the tension, the couplings are permanently fastened to the cable ends. This type of spliced joint also eliminates stiff places in the belt and provides for equal tension distribution from cable to cable.

Idler Selection

The second major item of cost in any high tension or long center belt conveyor application is that of the belt idlers. Ever since the firm establishment of the antifriction bearing type of belt idler, standard idler equipment has been fully equal to the demands of virtually every conveying condition. A wide range of good idler equipment is available today, and for high tension applications, it is only the high grade type of antifriction bearing idler that can be considered. Although all high grade idlers can be used for high tension belt conveyor work, there are some factors that must be considered.

A good seal is required to protect the bearings in applying idlers to high tension work. All high grade idlers have more than ample bearing capacity, and it is highly important that the bearings be kept protected so as to maintain this bearing capacity. For that reason the bearing seal must keep out grit and dirt effectively under all operating conditions. At the same time it must have a long life and not require periodic replacement.

A second important idler selection factor is the necessity of rugged roll construction. In order to have a rugged roll, the wall thickness must be adequate, and construction of the roll must be as a unit. There must be permanent connection of the shell to the ends and of the ends to the bearing tube or roll shaft. No joints should be present to loosen from impact and corrosion. The supporting brackets and the base on which they are mounted must be strong and rugged. If a strong roll construction is not adequately supported, the value of the roll design is largely lost.

In the usual appraisal of idler design, the first consideration is the antifriction bearing used, its capacity and life expectancy. This is only natural since the original replacement of the old plain bearing in idlers was due to the short life and high friction of the plain bearing. Now that the idler industry has had more than 25 years of experience with antifriction bearing idlers, that experience points to a different and more practical approach. Experience shows that roll wear, that is, shell wear, is actually the determining life expectancy factor for high-grade antifriction bearing idlers, where the bearing seals are good and the maintenance is adequate.

Roll Wear: Careful measurement was made of a number of rolls returned from the 10-mile belt conveyor transportation line used at Shasta Dam. These rolls were 5 in. in diam, with a No. 9 gage wall thickness to the shell, or approximately 5/32 in. The belt speed was 550 fpm and approximately 12,500,000 tons of aggregate were carried over a period of 3½ years, 2 years of which consisted of two 8-hr shifts per day, and 1½ years of one 8-hr shift. The amount of wear that could be detected on the bearings was so small as to be nonexistent, but on the roll shells, the maximum reduction in wall thickness on the rolls was approximately 5/64 in., or about half-way through the wall.

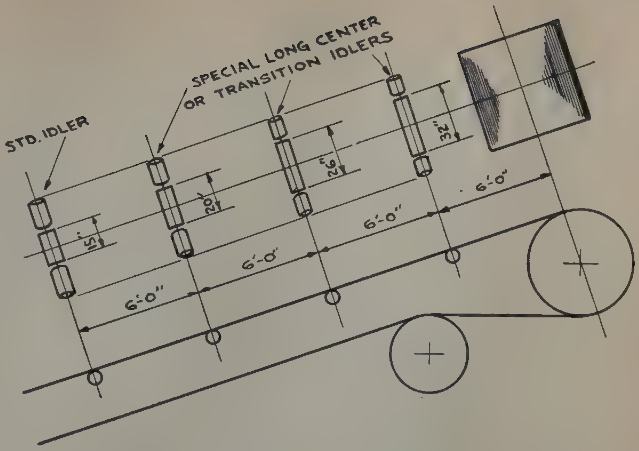


Fig. 6—Use of transition idlers for 42-in. high tension belt.

On several other occasions, close examination of bearings taken from idlers that had carried as much as 20 million tons also showed no measurable amount of wear. It becomes apparent that when protected by good seals and adequate maintenance, the life of the bearings in present day, high-grade, antifriction idlers, is not a controlling factor in idler life. Normal idler loads are not great enough to cause measurable bearing wear.

If bearing life in an antifriction bearing idler is virtually indefinite, then it can be ignored as a determining factor in idler life. Instead, it becomes apparent that the roll only must be considered. Further, assuming substantial unit construction, then only the life of the outer shell need be considered because the shell is the only part of the roll subjected to wear.

A number of operating factors affect the roll shell wear, either through the very slight abrading action by the belt in driving the roll, or the effect of shock and pounding upon the roll. Most important and obvious is belt speed, since that determines the number of contacts that the belt makes with the roll. The material load and weight also exert an important influence by determining the degree of pressure with which the belt contacts the idler roll. Where corrosion is present or abrasive dust is in the atmosphere, the wearing action of the belt on the roll is accelerated. Where idlers are tilted slightly forward to help belt training, some sacrifice in roll life is made, resulting from the slight additional skidding action of the belt contacting the roll.

Where large lumps are handled, the shock to the roll often results in deformation, which accelerates shell wear at the high points and correspondingly shortens shell life. However, the additional wear imposed by severe impact and shock is usually secondary since more serious damage results from deforming the supporting brackets, bending the roll shaft, or loosening the roll construction.

The importance of roll diameter to idler life has always been appreciated, but invariably it was the relationship of the rpm imposed upon the bearings by the diameter that was considered significant. With the determining factor of idler life being roll

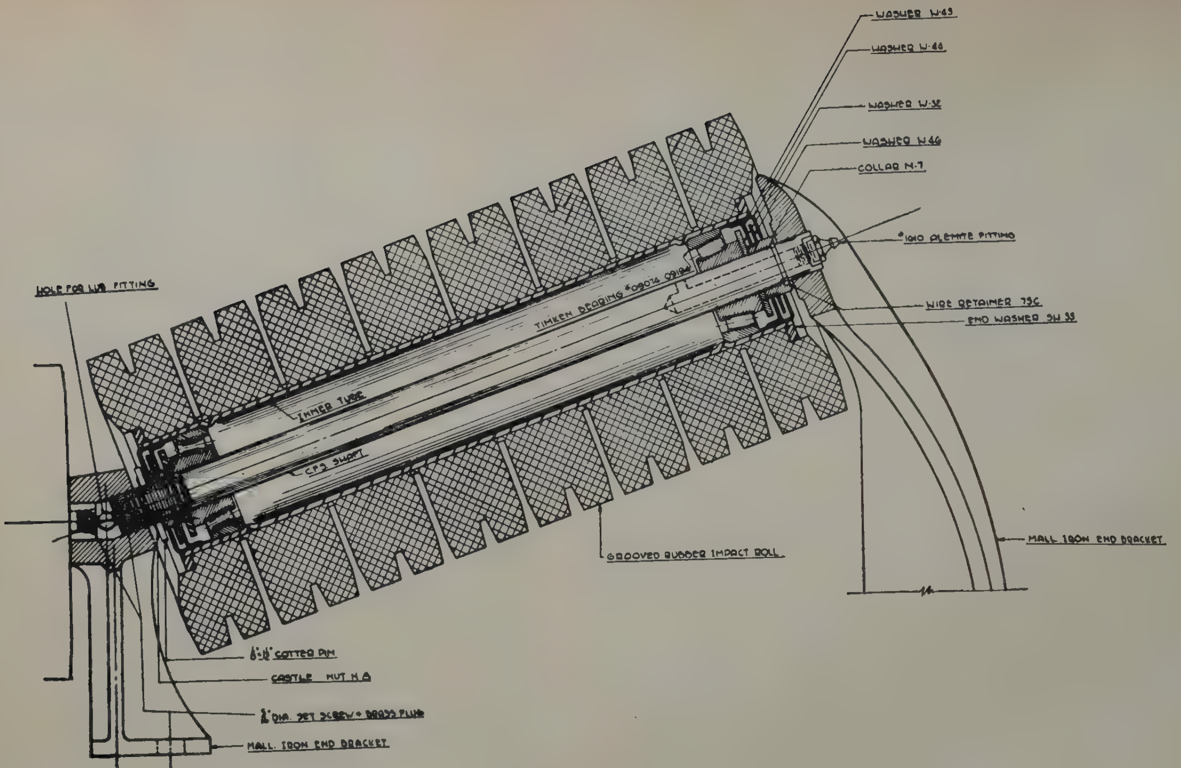


Fig. 7—Impact cushioning idler.

shell wear, the true importance of roll diameter obviously is its determining the number of belt contacts and not the number of bearing revolutions. The larger the roll, the slower the rpm and the fewer number of contacts per unit of time of the belt driving the idler roll. At a belt speed of 500 fpm, the roll rpms for various diameters are: 4 in., 394; 5 in., 318; 6 in., 265; and 7 in., 227. High grade antifriction bearing idlers are available in 5-in., 6-in., and 7-in. roll diameters, and if a 6-in. diam roll is taken as a normal, heavy-duty size, for a given belt speed that means the 4-in. diam roll makes approximately 50 pct more contacts with wear in direct proportion, the 5-in. diam roll approximately 20 pct more contacts and wear, while the 7-in. diam roll has approximately 15 pct fewer contacts and consequently correspondingly less wear. In addition, it is customary for the manufacturers to provide the larger diameter rolls with heavier walls, further penalizing the wear-life of the smaller diameter rolls.

Idler Selection: Since idler life expectancy is a function of roll shell wear, a logical approach to idler selection through the normal range of operating conditions becomes possible. Fig. 3 has been prepared as a guide for the selection of idler equipment for belt conveyor work. It is based upon actual experience of roll wear and the relation between roll diameters and wall thicknesses.

The selection of idlers for belt conveyors has always been a matter of experience rather than any clearly defined rules. One manufacturer made a step forward in simplifying idler selection by applying service factors to the operating conditions in order to determine whether the commercial grade ball bearing idlers should be used or the precision grade, antifriction bearing idlers. Fig. 3 goes further by including roll diameters, and if its use is tempered by experience, it should be a helpful selection tool.

In making idler selections, it has been only too common to give the carrying idler practically all of the attention with little consideration to the return idler. Actually, in all cases, and particularly for the high tension applications, the return idler must be given more attention than the carriers. The return idler rolls are in contact with the dirty side of the belt, so that there is a much more severe abrading action by the belt against the idler roll, resulting in accelerated reduction of the shell wall thickness through wear. It is universal experience that the return idler rolls must be replaced a number of times during the life of a carrier roll.

The return roll construction uses but two bearings located at each end with the roll spanning the complete frame, and deflection must be considered. After a time the belt takes something of a troughed set in its cross section, imposing localized pressure and contact at the roll ends rather than uniformly across the entire face of the return roll, accelerating shell wear at the ends. Moreover, the return idlers carry the belt where it is confined by the frame and difficult to see.

Considering the operating conditions for the return idlers, large diameter rolls should be used to insure maximum life by holding the number of contacts with the belt to a minimum. It is also advisable to provide the return rolls with heavier walls, which will further contribute to longer life.

While any high grade antifriction idler is basically suitable for high tension work, there are some considerations that must be given attention because of the combined effect of high speed and high tension in the belt. One of these factors is roll concentricity.

Roll Concentricity and Materials: For normal belt tensions and speeds, roll concentricity has seldom been a consideration. At high belt speeds, with high tensions, rolls that are seriously out-of-round can setup and impart very serious vibration. On one recent slope belt installation using a steel cable belt, some of the rolls that were slightly out-of-round, transmitted serious vibration to the steel gallery structure.

For high tension work, it would appear that the concentricity tolerance should be held to a maximum of 0.030 in. total runout on the diameter, particularly when the belt speed is 500 fpm and over. Although this tolerance is practical, it does call for close concentricity limits on the tubing used in fabricating the rolls.

For high tension belt conveyor applications, it is always desirable to provide the return rolls with an extra wide face. More than the usual provision must be made to permit lateral fleeting of the belt without danger of its running off the roll. This is accomplished by setting the frame stringers wider or farther apart, and adding proportionately to the face length of the return roll, usually an extra 6 in.

It is generally accepted practice for belt conveyor installations of all kinds to use either steel or cast iron rolls for the idlers. The high tension belt conveyor does not impose any new roll material requirement. In those instances where corrosion or excessive abrasion is present, the grey iron roll or a coated steel roll is a worthwhile investment.

Special Idler Applications: The spacing of idlers on the conveyor frame is an important cost consideration, since it determines the number of idlers, and therefore initial idler cost. This spacing must be close enough to prevent excessive sag of the belt between idlers, but at the same time, if the spacing is too close, the initial idler cost will be higher than necessary. Tables of average idler spacings based on experience and average conditions appear in the manufacturers' catalogs and handbooks.

In high tension applications, it becomes possible to tailor the spacing of the idlers very closely to belt tension, which simply means that as the tension increases, the idlers are spaced farther and farther apart. Knowing the belt tension, the weight of the loaded belt, and establishing a permissible amount of sag between idlers, the exact spacing of the idlers can be calculated, using the sag formula for a suspended cable. While there is difference of opinion as to the permissible amount of sag, a design figure of a maximum of ¼ in. per ft of span has given good operating performance.

The relationship of idler spacing to belt tension, with permissible sag limited to ¼ in. per ft of span, is shown in Fig. 4. It will be noted that the idler spacing is arbitrarily limited to a ceiling of 72 in., although theoretically a spacing of better than 9 ft is possible. Idler spacings beyond 72 in. have never been used, and as a matter of good practice this idler spacing should not be exceeded.

In designing the conveyor framework, it is more practical to space the idler by zones rather than to use a theoretical spacing for each idler. This obviously provides more economical frame construction, and there is not enough variation between the theoretical and the actual to have any effect upon operating results. Fig. 5 shows how the theoretical idler spacing in Fig. 4 has been converted into ten zones of spacing.

Graduated idler spacing is an important economic consideration since its use saves an appreciable amount of initial idler cost. The number of idlers required in Fig. 5 is 516. Using a conventional spacing of 3 ft 6 in., the number of idlers would be 683. Consequently there is a saving of 167 idlers through the use of graduated idler spacing, or, in this case, approximately 25 pct of the idler cost.

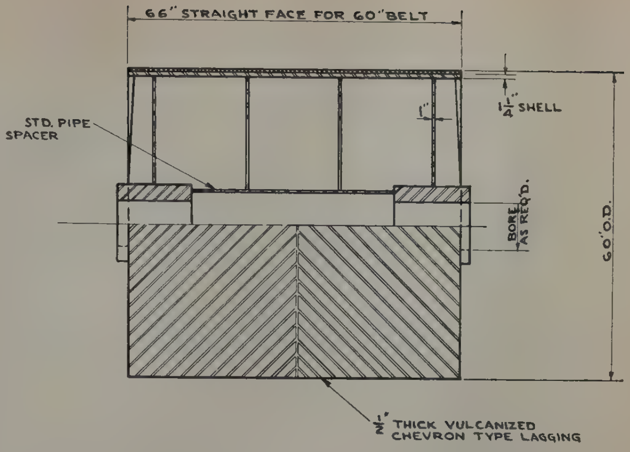

66" STRAIGHT FACE FOR 60" BELT
STD. PIPE SPACER
1"
1¼" SHELL
BORE AS REQ'D.
60" O.D.
½" THICK VULCANIZED CHEVRON TYPE LAGGING

Fig. 8—Typical welded steel drive pulley for high tension belts.

In high tension work it is particularly important that the change in contour of the belt in going from the troughed position on the idlers to the flat position over the head pulley must be done gradually over a considerable distance to keep the stresses in the belt edge from becoming excessive. The normal practice of not supporting the belt between the first idler and the head pulley is impractical because of the unusually long distance. Instead, a gradual change in belt contour is accomplished by means of special transition idlers in which the length of the center roll is gradually increased.

Fig. 6 shows the application of such transition idlers. The cost is nominal since it is possible for the manufacturer to use various standard lengths of rolls and supporting brackets to achieve the combinations indicated.

Although the use of impact cushioning idlers is important for any belt conveyor where lumps are being handled, such idlers are particularly important for high tension work in order to protect the very substantial belt investment. It is the function of the impact cushioning idler to cushion the blow on the belt on the idler roll resulting from lumps dropped on the belt, which it does by absorbing the energy of such impacts through momentary distortion of the roll. Adequate design of an impact cushioning idler must provide sufficient energy absorption capacity over a reasonable range of lump sizes and weights, together with exceptionally heavy and rugged supporting brackets and bases.

Several different designs of impact cushioning idlers are available; one very effective design is shown in Fig. 7.

The subject of training idlers, both carrying and return, is debatable. Theoretically, their use is unnecessary, for a good belt, properly spliced, operating on a well-constructed conveyor frame, can be trained to run perfectly true. Nevertheless, the use of training idlers is frequently desirable for no other reason than to correct unpredictable operating conditions which may develop after starting the system into operation.

Several types of construction of training idlers are available, but as yet none are completely satisfactory, and all of them in varying degrees are apt to cause belt damage. For the return self-aligning units it is particularly important that the belt be

Table V. Minimum Pulley Diameters—Inches

For High Tension Belts

COTTON CORD				STEEL CABLE			
No. of Plies	Drive, Head, or Tripper	Tail Takeup or High Tension Snub	Bend Pulley or Low Tension Snub	Class	Drive, Head, or Tripper	Tail Takeup or High Tension Snub	Bend Pulley or Low Tension Snub
5	42	36	24				
6	48	36	30				
7	54	42	36	1000	54	42	36
8	60	48	36	1500	60	48	36
9	60	54	42	2000	60	48	36
10	72	60	48	2500	60	54	42
				3000	72	60	48

High Strength Fabric

	STYLE 85 AND CX				STYLE CL			
No. of Plies	Tandem Drive	Single Drive, Head and Tripper	Tail Takeup High Tension Snub	Low Tension Snub and Bend	Tandem Drive	Single Drive, Head and Tripper	Tail Takeup High Tension Snub	Low Tension Snub and Bend
4	42	30	24	18	42	36	30	24
5	42	36	30	24	48	42	36	24
6	48	42	36	24	54	48	42	30
7	54	48	36	30	60	54	42	30
8	60	54	42	30	60	54	48	36
9	60	54	48	36	72	60	54	36
10	72	60	54	36	72	60	54	42
11	72	60	54	42	84	72	60	42
12	72	72	60	42	84	72	60	42
13	84	72	60	42	84	84	72	48

thoroughly cleaned since any build-up on the return roll is apt to completely defeat the guiding effect.

The construction of training idlers for high tension work must be exceptionally rugged, and generally speaking, standard units are inadequate. In order to obtain a proper steering effect from the training idlers, it may be advisable at times to link two or three such units together in clusters, together with some positive mechanical means of steering the belt.

If the rubber spiral or the rubber disk type of return idler rolls are used to provide additional wear resistance, care must be taken to see that these types of idlers are not located where the return belt is under high tension, as at the head end of slope conveyors where the drive is located at some distance back from the head. The present constructions of such spiral and disk type idlers do not hold concentricity within close enough limits to prevent imparting vibration to the belt.

Terminals

Although the terminal equipment does not represent as much of the total investment as the belt and the idlers, its selection is equally important. Included in the terminal equipment are the head, drive, snub, and tail pulley assemblies, takeup arrangements, as well as such auxiliary devices as belt cleaners.

Pulleys: For drive and high tension snub pulleys, the basic selection factors are those of adequate diameter and strength and grooved vulcanized lagging. The correct relation between pulley diameter and number of plies is given in the handbooks of manufacturers. With belts of multiple ply construction, this relationship is a function of the flex life of the carcass of the belt going around the pulley. With belts where the tension member is a single plane, and particularly with steel cable belts, the factor determining pulley diameter is that of face pressure of the belt against the pulley. Approximately 60 lb per sq in. is the usual maximum allowable face pressure on the pulley.

It is always better to make the pulleys larger in diameter than they need be, rather than smaller, since larger diameter pulleys pay off in better belt operation and longer belt life. But with high tension applications, the head and drive pulleys are of heavy, expensive construction, and consequently there is a cost limitation to making pulleys unnecessarily large in diameter. The various belting manufacturers have all established minimum diameters of pulleys for the several belt constructions which they manufacture, and Table V illustrates several such typical minimum pulley diameter recommendations.

It is most important for high tension belt conveyor applications that the pulleys have great strength with ample factors of safety to withstand the tremendous crushing effect of the conveyor belt, under extremely high tension, wrapped around it. Standard conveyor pulleys, both the welded steel and cast-iron types, are designed for 120 lb per in. of width belt tension, and such pulleys, when furnished as heavy-duty pulleys, have a tension rating of approximately 200 lb per in. of belt width. In high tension work, the belt tensions per inch of width generally start at approximately 500 lb and can be as high as 3000 lb, so it is immediately evident that standard commercial conveyor pulleys are completely inadequate for high tension applications.

The usual design of a high tension pulley is a very heavy, welded steel type of pulley with reinforcing diaphragms. The shell or wrapper is formed from heavy steel plate, as are the diaphragms, with plate thicknesses ranging from ½ in. to as much as 2 in. Such pulleys are completely stress-relieved after welding, and then are machined to size. A typical welded steel drive pulley is shown in Fig. 8. Some very heavy cast-iron, high tension conveyor pulleys have also been used. A welded steel pulley of dead shaft design is shown in Fig. 9. In every case, the pulleys are carefully designed for the specific application.

The type of rubber lagging used on the drive pulley is a grooved lagging of herringbone design. Rubber lagging the drive pulley is a must in high tension work in order to provide the maximum coefficient of friction between the pulley and the belt. Further, it is necessary that this rubber lagging be the grooved type as shown, in order to maintain the high coefficient of friction even if the belt should become wet. Laboratory tests confirmed by field results have demonstrated that the grooved type of lagging has the same coefficient of friction when wet as a smooth lagging when dry, and it is the latter that is used for conveyor design.

Because of the grooving, this type of lagging cannot be applied by bolting as is commonly done with smooth rubber lagging, but, instead, it is vulcanized directly to the conveyor pulley face. The lagging is built up ply by ply right upon the pulley face, and then the whole assembly cured at high temperature for the vulcanizing process. Although the rubber manufacturers have been doing this regularly for the pulley builders, the handling and curing facilities should be checked because of the massiveness of the pulley construction and assembly, which can easily run to 10 tons or more in weight.

In addition to the factors of diameter, strength, and lagging, there is another special consideration

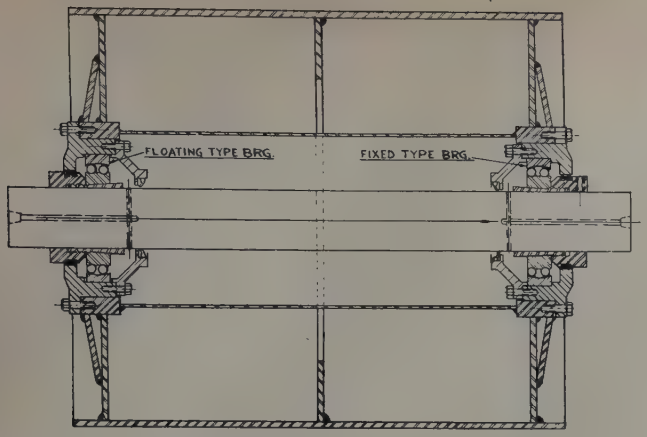


Fig. 9—Welded steel pulley, dead shaft design.

for high tension pulleys. Normally conveyor pulleys are made with a crown, but for high tension belt applications, the pulleys are made with a straight face. Because of the extremely high tensions in the belt and the necessity of distributing this tension uniformly across the entire width of the belt, the use of any crowning is harmful to the belt. This construction has not imposed any operating difficulties, since with high tensions it is easier to keep the belt trained on the straight-face pulley.

One further practical consideration in designing pulleys for high tension work is making the face wider than normal. It has been customary to make the face of the pulley approximately 2 in. wider than the belt for the average size belt, and 3 in. wider for belts of 48 in. and over. However, for high tension applications, the pulley face should be at least 6 in. wider than the belt and occasionally even more becomes desirable. This extra face width is simply to make sure that the belt never runs off the pulley to one side or the other. In spite of everything that is done in training the belt, at times there may be some fleeting on the head pulley, and under conditions of very high tensions, if the edge of the belt actually ran over the edge of the pulley face, serious belt damage could quickly occur.

Where pulleys are located at points of low tension in the belt, standard, heavy-duty design pulleys are used. This always must be checked since it is poor economy to use pulleys without ample factors of safety. There have been many instances where pulleys have had to be replaced with heavier designs after short periods of operation in the field.

Shafting: Shafting for high tension belt conveyor work, involves no special design other than sound, conservative engineering. Because of the larger diameters required, the usual material is forged steel, which occasionally will need to be high carbon or alloy. The design of the shaft is based on combined bending and torsion, and if high carbon or alloy shafts are used, the size must be carefully checked in order to avoid excessive deflection.

Bearings: For high tension belt conveyors, practical considerations dictate that antifriction bearings be used exclusively. With antifriction bearings, the original installation becomes simpler and easier than if sleeve bearings are used, and the customary run-in period of sleeve bearings, with the messy greasing conditions during that run-in period, is entirely avoided. Moreover, the antifriction bearings require only nominal service attention, and they secure the shafts against any lateral float. While the initial cost of the antifriction bearings themselves is higher, there will not be a great deal of difference in the final installed cost, and what cost difference may exist is completely offset by the operating advantages.

Takeup Arrangements: Long center belt conveyors require careful study of the takeup problems, and without exception, some form of a gravity takeup must be used in order to definitely control and provide the correct slack side tension for sufficient driving effort under all conditions. The amount of takeup travel necessary and the location of the takeup pulley in its travel must be considered in addition to providing the required counterweight.

The amount of takeup travel is determined by two considerations, the amount to provide for belt stretch, plus an additional amount to provide for vulcanized splicing of the belt. It is obvious that the amount of travel to provide for belt stretch will be determined by the stretch characteristics of the belt construction used, and the amount necessary can range from almost 2 pct on short center fabric belts to as little as 0.025 pct of the center distance for steel cable belts.

Belt Stretch Allowance: It is obvious that the stretch in a belt will be proportional to the tension in it and that the stretch in the belt is not uniform over its entire length since the tension is not uniform. Consequently, as conveyor centers increase, the amount of takeup movement necessary in percentage of the total centers becomes less. In addition to the effect of tension, temperature and atmospheric conditions also exert influence upon belt stretch.

In addition to providing for belt stretch, a very generous allowance must be made to provide enough slack in the belt to make the vulcanized splice. This is important in order to provide plenty of working length to make the final splice or for any subsequent splicing that may be necessary. The amount of takeup movement to allow for splicing preferably should be sufficient to make two vulcanized splices, or at least 10 ft. An extra 2 or 3 ft will invariably prove to be an excellent investment.

Initial Position of Pulley: The initial position of the takeup pulley in its travel is important for the

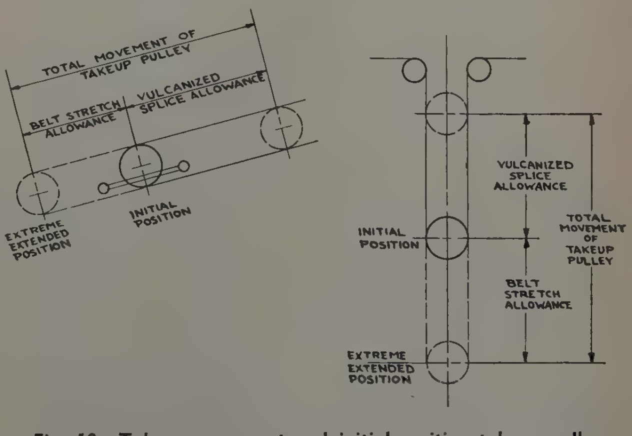

TOTAL MOVEMENT OF TAKEUP PULLEY
BELT STRETCH ALLOWANCE
VULCANIZED SPLICE ALLOWANCE
EXTREME EXTENDED POSITION
INITIAL POSITION
VULCANIZED SPLICE ALLOWANCE
INITIAL POSITION
TOTAL MOVEMENT OF TAKEUP PULLEY
BELT STRETCH ALLOWANCE
EXTREME EXTENDED POSITION

Fig. 10—Takeup movement and initial position takeup pulley for high tension belts.

Total movement of takeup pulley, ft = movement back or down from initial position, ft, for belt stretch allowance plus movement forward or up from initial position, ft, for vulcanized splice allowance.
Belt stretch allowance, ft = initial movement allowance, ft, Table VI, X exposure factor, Table VI.
Initial position of takeup pulley = belt stretch allowance, ft, forward or up from extreme extended position.

high tension belt conveyor. Ordinarily the takeup pulley is located somewhere close to the forward position in the takeup so as to allow most of the movement for belt stretch. With a high tension belt application, however, it must not only be possible for the takeup pulley to move back or downward to take care of belt stretch, but at the same time the takeup pulley must be able to move forward or up in order to provide the slack for vulcanizing the belt. The use of Tables VI and VII and Fig. 10 will not only determine the amount of travel to take care of belt stretch under various conditions but will also give the initial position of the takeup pulley. The results from these tables will provide adequate amounts of takeup movement without setting up an excessive amount of movement on conveyors of very long centers.

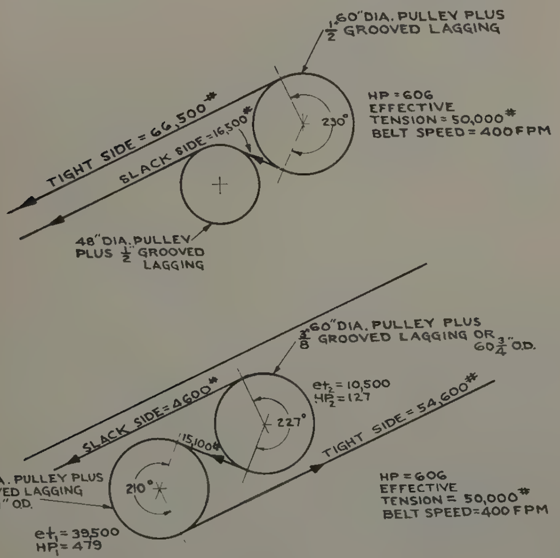
Gravity Takeup Application: The ideal location for a gravity takeup is at the point of lowest belt tension, for at that point the least amount of counterweight will be necessary. On a horizontal conveyor that would mean placing it immediately behind the drive pulley, on an inclined conveyor at the tail pulley, and on a declined conveyor at the discharge pulley. If the takeup cannot be located at points of low belt tension, then it will be necessary to use a corresponding amount of increased counterweight to make it operative at the point of location.

Fig. 10 shows schematically the difference in action between the horizontal gravity takeup and the vertical type. In the design of vertical units, enough height must be provided for the total movement of the takeup pulley, and it is important that the movement of the takeup pulley be in guides so that there will be no adverse effect upon the training of the belt on its return run. With the horizontal gravity takeup, the carriage should operate on inverted angle tracks which will be self-cleaning, and the wheels as well as the pulley should have antifriction bearings in them. Where great amounts of counterweight are required, dual diameter cable drums or other arrangements of reaving the takeup weight cable may be used to reduce the actual weight used.

Fig. 11—Example of tight side tension for snubbed pulley and dual pulley drives with division of load on dual pulleys.

$$\frac{66{,}500}{\text{48-in. belt}} = 1380 \text{ lb tension per in. of width}$$

$$\frac{54{,}600}{\text{48-in. belt}} = 1140 \text{ lb tension per in. of width}$$


60" DIA. PULLEY PLUS $\frac{1}{2}$" GROOVED LAGGING
TIGHT SIDE = 66,500#
SLACK SIDE = 16,500#
230°
HP = 606
EFFECTIVE TENSION = 50,000#
BELT SPEED = 400 FPM
48" DIA. PULLEY PLUS $\frac{1}{2}$" GROOVED LAGGING
60" DIA. PULLEY PLUS $\frac{3}{8}$" GROOVED LAGGING OR 60$\frac{3}{4}$" O.D.
SLACK SIDE = 4600#
15,100#
227°
et_2 = 10,500
HP_2 = 127
TIGHT SIDE = 54,600#
60" DIA. PULLEY PLUS $\frac{1}{2}$" GROOVED LAGGING OR 61" O.D.
210°
et_1 = 39,500
HP_1 = 479
HP = 606
EFFECTIVE TENSION = 50,000#
BELT SPEED = 400 FPM

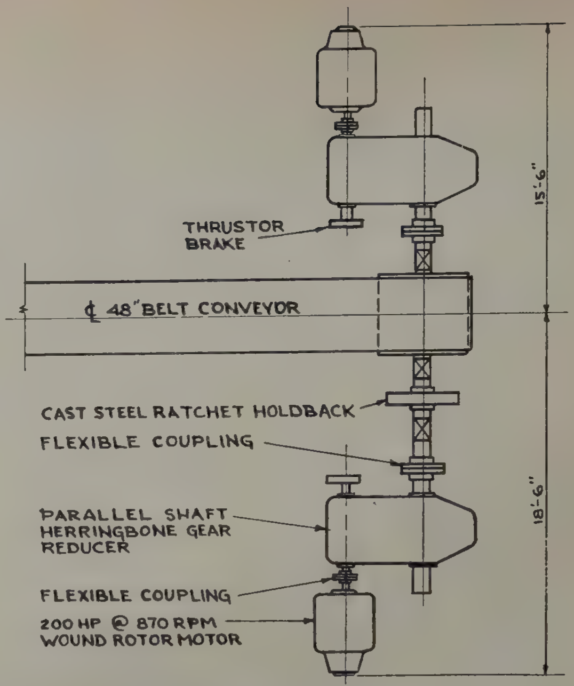

15'-6"
THRUSTOR BRAKE
℄ 48" BELT CONVEYOR
CAST STEEL RATCHET HOLDBACK
FLEXIBLE COUPLING
PARALLEL SHAFT HERRINGBONE GEAR REDUCER
FLEXIBLE COUPLING
200 HP @ 870 RPM WOUND ROTOR MOTOR
18'-6"

Fig. 12—Snubbed pulley dual motor drive arrangement for high tension belts.

Belt Cleaners: Mention must be made of the belt cleaning problem, because no completely satisfactory cleaning device has yet been built. The common cleaning devices consist of various kinds of scrapers, wipers including spiral wipers, slow and high speed rotary brushes, electric vibrators, and high pressure water sprays. Each has its disadvantages, and the final choice of what to use will depend upon an individual study of the particular cleaning problem.

A belt with material adhering to it can impose serious operating problems on its return run, and in one case, serious study is being made of the possibility of introducing a 180° twist into the conveyor belt at each end of the return run so that the clean side of the belt will be in contact with the return idlers. Where space conditions permit, such a scheme would appear to be practical.

Drives

The tremendously expanded horsepower capacity of the modern, high tension belt now makes drives of hundreds of horsepower a commonplace for belt conveyors, and the record to date is chalked up by an installation now being made which will use a 1500 hp motor. The selection and application of such large horsepower drives for belt conveyors calls for careful study.

Pulley Drives: The drive arrangement for high tension belts is very often made simple by taking full advantage of the high tension capacity of the belt and using a simple snubbed pulley drive arrangement at the head pulley. Applying all of the horsepower to a single snubbed pulley is much preferred to using a dual pulley or tandem drive, which introduces problems of its own. Fig. 11 shows the belt tensions when using a snubbed pulley drive and a dual pulley drive for the same problem. In this case, while the snubbed pulley drive arrangement does impose 12,000 lb additional belt tension, the maximum still falls within the working limits of the belt that would need to be used anyway. The 1140 lb per in. tension in the dual pulley arrangement would require the use of a 1400 lb or 1500 lb per in. steel cable belt, and the 1380 lb tension for the snubbed pulley arrangement still falls within the limits of that belt. This example is by way of illustration only since in the actual determination of the maximum belt tensions for the final selection of the belt, other considerations enter in. Even if a stronger belt must be used, the additional cost of the belt will usually be less than the added cost of the dual pulley drive arrangement.

One disadvantage of the dual pulley drive is the fact that one of the driving pulleys works against the dirty face of the belt, and particular care must be taken to keep the belt completely clean at all times to prevent the coefficient of friction between the belt and the driving pulley from being seriously lessened. It is possible to introduce additional pulley assemblies to be used as special snub pulleys between the drive pulleys, by means of which both the dual pulleys can be made to drive against the clean side of the belt. Such arrangements add to the initial cost of the installation and considerably complicate the drive arrangement.

When it is impossible to locate the drive at the head pulley, or for good reason all the power cannot be applied to a single pulley, the dual pulley drive arrangement is used very successfully. The design of such drives is fully covered in several handbooks and must be carefully worked out. The normal division of the load on a dual pulley drive is about as shown, although it is perfectly feasible and practical to divide the load between the two pulleys arbitrarily as may be desirable.

It should be noted that the secondary pulley in a dual pulley arrangement should be slightly smaller in diameter than the primary pulley. There are two reasons for this: (1) To keep the tension plane in the belt itself properly centered, inasmuch as there is a difference in the thickness of the carrying cover and the cover on the back of the belt, for example, 3/16 in. as compared to 1/16 in.; and (2) because there is an actual difference in the speed of the belt in going around the two pulleys, with the speed going around the secondary pulley slightly less because of the reduced tension in the belt. A slight difference in the pulley diameters compensates for these two factors.

Table VI. Initial Movement Allowance for High Tension Belt Takeup Travel

Center to Center Distance of Conveyor, Ft	CENTER DISTANCE, PCT			MOVEMENT ON CENTERS, FT		
	Fabric	Nos. 50, 70, 100 Cord	Steel Cord	Fabric	Nos. 50, 70, 100 Cord	Steel Cord
50				4	4	5
150	2.9	2.9		4.50	4.50	5
250	2.8	2.8		7	7	5
375	2.7	2.7		10	10	5
500	2.6	2.6		13	13	5
625	2.5	2.45		16	15	5
750	2.5	2.3		19	17	5
875	2.5	2.05		22	18	5.50
1000	2.5	1.8	0.65	25	18	6.50
1200	2.5	1.8	0.55	30	22	7
1400	2.5	1.8	0.5	35	25	7
1600	2.5	1.8	0.5	40	29	8
1800	2.5	1.7	0.5	45	31	9
2000	2.5	1.5	0.45	0.50	32	9
2200		1.5	0.45		33	10
2400		1.5	0.45		36	11
2600		1.5	0.45		39	12
2800		1.5	0.45		42	13
3000		1.5	0.45		45	13
3200 and over		1.3	0.4		0.013X Center Dist.	0.004X Center Dist.

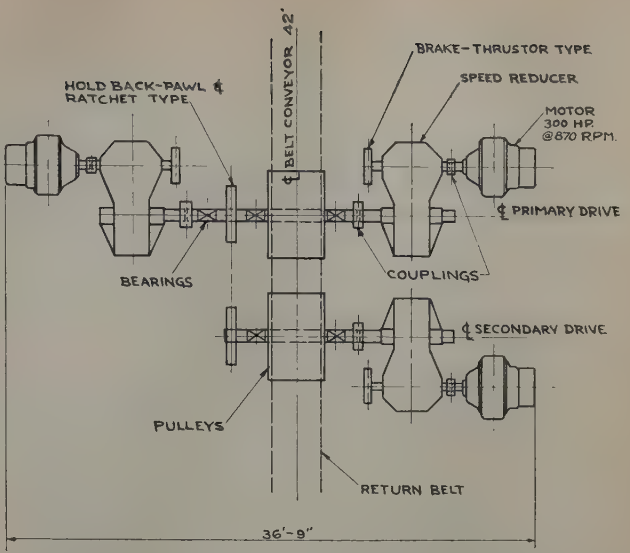


Fig. 13—Dual pulley multiple motor drive arrangement for high tension belts.

Direct Connected Speed Reducer Reductions: While it is entirely possible to use a secondary drive reduction such as a chain drive between the speed reducer unit and the drive shaft, generally it is more practical to couple the slow speed shaft of the speed reducer directly to the pulley drive shaft. The amount of detailed study and consideration that must be given to the high tension belt installation very closely determines the final belt speed, and any future variation in speed is unlikely. The fixed ratio of the speed reducer is not a handicap. Moreover, because of the high horsepower and the relatively slow speed, any secondary drive reduction must be massive in size and often the necessary space is not available. It is important to have the drive reduction as compact as possible, completely enclosed, and operating in oil.

In the large horsepower and the large reduction sizes, the range of selection of standard speed reducers now available is rather narrow with big jumps between sizes. It is therefore frequently necessary to select a reducer unit that is larger in capacity and of higher initial cost in order to obtain one of sufficient capacity.

Motor and Tail Drives: Consequently, it frequently may be possible to achieve initial cost economy through the use of two smaller drives rather than a single large one. While such practice entails care in the selection of the motors and the control equipment, dual and multiple motor drives are entirely practical and such drives are in use. Illustrating what can be done, Fig. 12 shows the use of two motors coupled to a single snubbed pulley drive shaft, while Fig. 13 shows three drives of equal capacity applied to a dual pulley arrangement. With multiple drives, difficulty with any one of them does

Table VII. Exposure Factor for High Tension Belt Takeup Travel

Conveyor	FABRIC BELTS		CORD BELTS	
	28, 32 and 36 Oz.	48 and 42 Oz.	Nos. 50, 70, 100	Steel
Standard or Outside (Covered), pct	80	60	60	40
Outside Not Covered, pct	75	55	55	40
Underground, pct	70	50	50	40

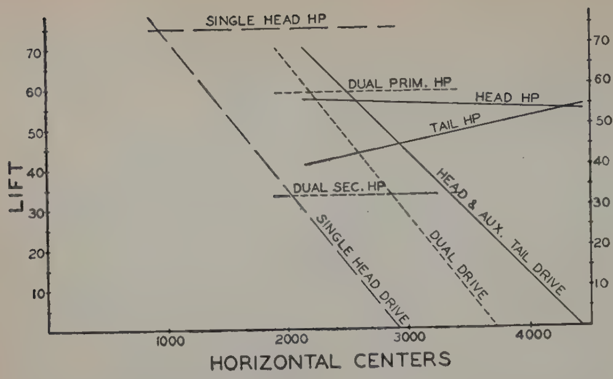


Fig. 14—Comparison of conveyor centers and horsepower. Proposed 5-mile belt conveyor haul, 5 ply 42 oz. 36-in. belt, 350 fpm, 600 tons per hr.

	No. of Flights for 5 Miles	Ft per Flight Horizontal
Single head drive	9	2920
Dual head drive	7	3690
Head and auxiliary tail drive	6	4390

not completely shut down the conveyor, which can continue to operate at reduced capacity. There is also the advantage of economy in the carrying of spares.

In laying out transportation by means of belt conveyor systems, an auxiliary drive can be used at the tail end of individual conveyors in order to convert more of the tension capacity of the belt into working tension. The net effect is to make longer centers possible, with the use of fewer units and therefore a reduction in the number of transfer points. Locating a drive pulley at the tail is simply widely separating a dual pulley drive, moving the secondary pulley to the tail. Doing this eliminates some of the inherent disadvantages of the dual arrangement. A better natural division of load is achieved, the need to slightly vary the pulley diameters disappears, and it is relatively easy to have the clean face of the belt also contacting the tail or secondary drive pulley. The effect of a tail pulley drive on increasing conveyor centers is shown in Fig. 14, which also includes the similar but limited effect of a dual pulley drive. The cost of the greater number of smaller horsepower drives, together with the increased cost of controls and wiring, must be balanced against the advantages of eliminating transfer points, to determine whether or not such an arrangement is economically desirable. Applying a drive at the tail end of a conveyor presents no particular mechanical problem and Fig. 15 shows one possible arrangement.

Motors and Controls: The proper selection of the motors and controls is a problem for the electric equipment manufacturers to solve. Several considerations must be called to the attention of the electrical equipment supplier. Because of close control of the starting torque and the balancing of motor speeds, the wound rotor type of motor is a must, and where dual and multiple motors are involved, they must be carefully matched. On declined units, the winding must be suitable for generation of power by the motor, which thus serves as a brake to govern speed and prevent runaway. Generally it will be more economical to use low-speed motors instead of using larger reductions through the speed reducer units.

With the controls, endless combinations can be worked out, and careful study must be given to the entire control problem. They must provide for delayed or stepped starting, building up blocks of torque for easy breakaway, but must add blocks slow enough during the acceleration period so as not to exceed the amount of torque required for breakaway. With dual or multiple motor arrangements, resistances for balancing motor speeds to the loads must be heavy duty because such motors will always be running on resistance.

With inclined conveyors the controls must include a time delay in the holdback circuit to keep the brake in the set position until the motor has had time to build up sufficient torque to prevent runback. On inclined conveyors, a zero speed switch arrangement is used to set the brake immediately in the event of any mechanical failure in the drive. Where a system of conveyors having interlocked sequence starting is involved the reverse sequence stopping is a must. Provisions for emergency stopping is important. This can be worked out best by running an emergency stop wire or cord the full length of the conveyor enabling emergency stops from any point or by placing a number of emergency stop buttons along the conveyor. A single start station must be used at the head of the conveyor, and an operation procedure must be worked out to prevent restarting until the cause of the emergency stop has been cleared.

Speed Reducers: Except for selecting the most economical size, there is no special problem in applying speed reducers to high tension belt applications. The range of selection in the high horsepower units is narrower than is desirable, and it is hoped that this condition can be corrected by the speed reducer manufacturers. The speed reducers used must be of the highest grade possible and adequately oil lubricated.

The reducer for dual or multiple motor drives must be capable of carrying heavy overloads for short periods of time. In multiple arrangements when one motor cuts out, the remaining drive or drives must then be able to operate overloaded long enough to empty the belt. Conservative selection of the speed reducer equipment is highly important for high tension belt applications.

The couplings that are used must be picked equally conservatively and must be completely dependable. For this heavy duty work the enclosed lubricated gear type of coupling has proved to be most successful, although other types of couplings also have been used.

Holdbacks or Brakes: With inclined or declined belt conveyors the holdback device or brake is an important item of equipment. The runback of a long, loaded inclined conveyor, or the runaway of a declined, loaded conveyor can be very disastrous.

Any of the customary types of holdback devices can be used for high-tension applications, band brakes, ratchet and pawls, roller holdbacks, or thrustor brakes on the motor or reducer shafts. It is of prime importance to see that the holdback device selected is far oversize in capacity. Very conservative selection of the holdback is the cheapest kind of insurance.

Where band brakes are used, the brake lining must be held out of contact during normal rotation, and the high horsepower capacity calls for post brake or hoist design. The slope conveyor of 1500 hp previously mentioned uses a regular mine hoist brake.

Where a ratchet and pawl arrangement is used, the ratchet wheel must be located on the drive shaft, and the pawl held out of engagement during normal rotation by means of a thrustor. Interruption of current for any reason, stopping, power failure, or by a zero speed switch, immediately drops the pawl.

Conservative practice dictates the use of two holdbacks, a thrustor operated brake on the extended motor shaft or high speed shaft of the speed reducer for all normal stopping requirements, and a mechanical brake on the drive shaft to become operative should there be a mechanical failure of any kind in the drive train.

Declined conveyors introduce problems of their own, inasmuch as there is no holding against reversal of rotation and runback, which immediately rules out the use of any ratchet and pawl or roller holdbacks, and requires the use of some form of band brake arrangement. For declined conveyors, a very generous oversize rating selection with adequate provision for heat dissipation is a must.

Conveyor Frame and Supports: The design of the conveyor frame and its supports for the high tension application does not introduce particular design considerations, although it should be obvious that with the high machinery investment involved, it would be poor economy to skimp on the design of the frame and supports. When located in the open, complete housing for the conveyor must be provided in order to adequately protect the investment in the belt and the machinery.

The frame should be made wider than would be normal, with the distance between the stringers equal to at least the belt width plus 14 to 15 in., and in some cases possibly more. A unitized type of head frame construction in which all mountings for bearings, reducers, and motors are machined after welded assembly, and which insures perfect alignment of the drive elements, is also to be highly desired.

Erection and Operation: The problems of the high tension belt conveyor during the installation of the equipment stem from the massive proportions of the heavy machinery items that must be handled. Careful planning is always necessary, and frequently the problem of transporting the heavy terminal pulley assemblies, the large rolls of conveyor belting, and the heavy pieces of drive machinery, particularly where this must be done under ground, may in itself be the determining factor in dictating the use of a single flight slope conveyor, instead of two or three sections for which a much less expensive belt could be used. The problem of transporting under ground frequently exerts definite limiting effect upon the size of the terminals that can be used.

The problem of reaving the conveyor belt into place and making the necessary vulcanized splices is also one that cannot be treated casually. Inasmuch as several splices are required, adequate time for the splicing must be figured. Including the preparation of the belt ends, which takes far more time than the curing, generally from one to two days' time is required for each splice that is made.

After the belt has been made endless, it must be trained so that it will run true first empty, then full. Even with the most accurate initial setting of all of the idlers, the training period will still require individual adjustment of a large number of the idlers, both carrying and return. Training or aligning idlers should never take the place of proper training of the belt. One operator follows the very sound practice of insisting that his men properly train the belt before he permits the placing of any training idlers in the conveyor.

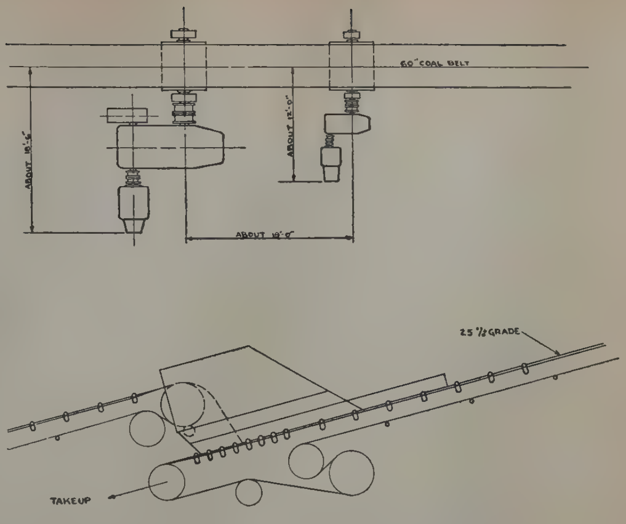


Fig. 15—Auxiliary tail pulley drive arrangement.

It is important that maintenance routines be immediately established for proper care of the belt. All cuts and any edge damage must be repaired immediately. While it is seldom possible to shut down long enough to make a complete repair whenever damage is first noticed, it is important to shut down long enough to apply rubber cement to the cut or damage, which will afford protection until the repair can be properly made and vulcanized during an off-shift period. The special cements and adhesives used in repairing belts deteriorate with age, so the supply department must regularly furnish fresh quantities, regardless whether the previous lot has been used or not.

In connection with greasing the idlers, when it is understood that the primary function of grease in a belt idler is not that of lubrication but rather a means of sealing out grit and dirt from the bearing, then it becomes clear that an infrequent greasing cycle is completely adequate. A minimum greasing cycle of approximately 1000 hr of operation is generally conservative, and observation frequently will show that such a cycle can be lengthened. The one exception is operation under extremely wet conditions. Moisture and water tend to emulsify and drain grease from the idler bearing, so that under such operating conditions, frequent renewal of the grease is necessary. Where water conditions are particularly bad, a water repellent type of grease should be used.

The very best evaluation of the recommendations of the belting and equipment suppliers, and the most thorough engineering and cost study of a high tension belt conveyor installation seldom completely covers every last detail that is encountered. Appropriate contingency items, reasonably generous, should be placed in the project budget.

Acknowledgments

Grateful acknowledgment is made of the helpful criticism of S. D. Michaelson, Chief Engineer of Coal Mines, Tennessee Coal Iron and Railroad Co., whose opinions are highly regarded because of his operating experience with one of the outstanding slope belt conveyor installations. Use has also been made of available material, published by conveyor equipment and conveyor belting manufacturers.

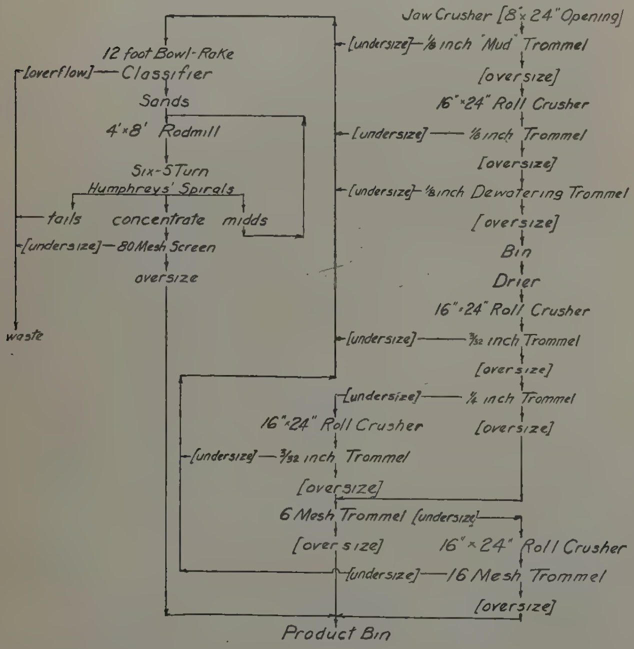
New Method for Recovery of Flake Mica

by R. Adair, W. T. McDaniel, and W. R. Hudspeth

A NEW method for concentrating the flake mica either from present washing plant tailings or from new feed has been developed. In this paper, flake mica refers to that which occurs in weathered granites, alaskites, pegmatites, and schists in a form too fine to be considered sheet mica. It ranges in size from 1 in. or more down to 200-mesh or finer and occurs with kaolin, quartz, and partially kaolinized feldspar. Scrap mica, a term often used indiscriminately to describe everything other than sheet mica, will be used to denote the waste product from sheet mica mines and fabricating plants. Both flake and scrap mica are used in the mica grinding plants to produce wet-ground or dry-ground mica, although some flake mica recovered from clay mining operations is fine enough to be marketed without grinding.

Until 1930, ground mica was made almost entirely from scrap. From 1930 until 1935 a gradually increasing amount of flake mica was mined, and most of this was sold without grinding. A large percentage of this mica was recovered from clay washing plants. In this type of operation the recovery of any sheet mica was of secondary importance and was considered a byproduct.

Fig. 1—Typical mica washing plant revised to treat tailing on Humphreys spirals.



North Carolina produces more flake mica than any other state, accounting for 73 pct of the total domestic production.[1] In the following review of current practice in jig plants, there is no intention to disparage these operations. Within the limitations under which they were built, they are examples of mechanical ingenuity. The term jig plant is a carry-over from early practice since there are no jigs in operation at the present time. Current practice, as shown in Fig. 1, is based on successive stages of differential crushing with rolls followed by trommel screens to remove the more easily crushed gangue minerals. The oversize mica product then is usually dried, and passed through another set of dry rolls and a final screen to remove the remainder of the grit. This process is effective in recovering up to 50 pct of the mica in the original ore, the balance being lost in the screen undersize, usually —⅛ in. Jigs and tables were at one time used to treat this screen undersize and were capable of recovering a part of the mica larger than about 60-mesh.[2] Fine mica lost in these plants is normally washed away in streams and can be considered irrecoverable.

There are ten or more washing plants in the North Carolina area that are losing approximately 50 pct of the mica in the feed. Since the three cooperating agencies, the Asheville Minerals Research Laboratory of North Carolina State College, the North Carolina Dept. of Conservation and Development, and the Tennessee Valley Authority, are concerned with conservation of minerals, they have investigated means for improving the efficiency of these mica plants. The mica industry has been cooperating fully with the laboratory to establish a more efficient method of recovery for flake mica.

Choice of Methods

The initial investigation was concentrated on finding a means of reducing the losses from plants now using the conventional flowsheet shown in Fig. 1. To accomplish this it would be necessary to collect the screen undersize products through the plant and remove from them the mica from ⅛ in. down to

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150-mesh or finer. Since the size of any of these operations is small and capital limited, the method had to be one which could be set up at moderate cost and operated by the present labor force.

Flotation, tabling, and jigging were considered, but before starting work on these, a test was run on a Humphreys spiral, which gave evidence of a definite separation of the mica from the gangue minerals. As compared to other gravity methods or flotation it appeared to offer the following advantages: (1) lower capital cost than flotation or jigs and tables; (2) less skilled labor required; (3) lower operating cost.

Since the ore is relatively cheap and abundant, recovery is not of first importance. Later tests proved that flotation would give the best metallurgical results. The spiral was rated next in efficiency followed by jigs and tables.

Concentration By Humphreys Spiral

This separation is based on shape difference, whereby the thin flakes are carried to the outside of the stream and the more compact particles of gangue are drawn down the inner ports. Mica would move toward the ports were it not for the shape difference, since its specific gravity is slightly higher than that of the gangue.

The important variables tested were feed rate, percent solids in feed, amount and distribution of wash water, degree of grinding of the feed, and extent of middlings recirculation. Analysis of the samples taken from spiral tests was made by separating them by flotation into two or more products whose mica content could be accurately appraised by examination with a binocular microscope. For example, a sample containing 50 pct mica would be very difficult to appraise, but it would be relatively easy to estimate the gangue in a +95 pct mica concentrate or the mica in a clean tailing. Some fine fractions were analysed by means of a heavy-liquid separation in a high-speed centrifuge. The slightly higher specific gravity of the mica caused it to report in the bottom of the tubes. Flowsheets for Humphreys spiral tests are given in Fig. 2.

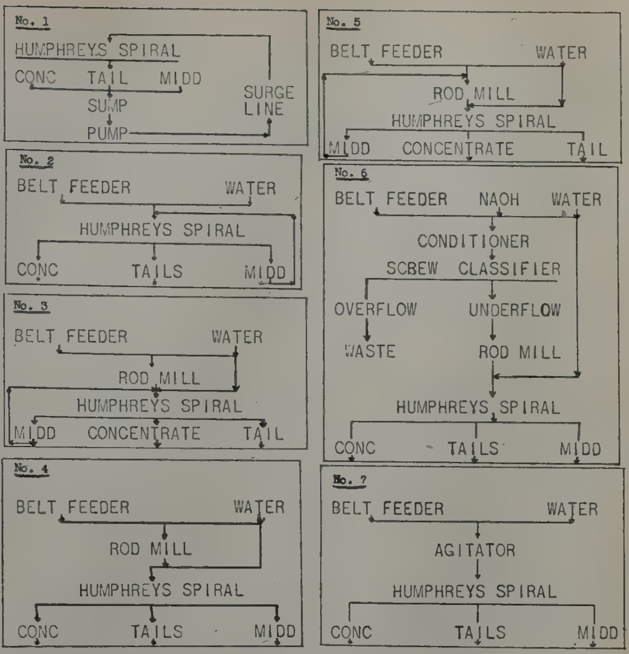


Fig. 2—Flowsheets for Humphreys spiral tests.

In general, the extent of feed preparation necessary for efficient spiral separation depends on the natural degree of delamination of the mica flakes. Most samples tested responded poorly to direct spiral treatment, but one exception to this rule makes further investigation necessary on the subject, Table I, Test 1, with a concentrate grade of only 45.8 pct mica, as compared with Tests 12 and 13 with concentrate grade up to 91 pct mica. In this case, since the mica is already in thin flakes, it is feasible to concentrate it directly. Since the spiral concentration of mica is based solely on shape difference, most ores require a pass through some form of reduction equipment to delaminate the mica and accentuate its shape difference. The feed preparation should be

Table I. Test Data

Test No.	Lab. Sample No.[a]	Flowsheet[b]	Feed Preparation	New Feed, Lb per Hr	Solids in Rod Mill, Pct	Circulating Load, Pct	Spiral Feed: Lbs per Hr	Spiral Feed: Solids, Pct	Recov. Mica, Pct	Mica in Sp. Conc., Pct	Mica in Feed, Pct
1	649	1	none				682	10	72.6	45.8	15.0
2	650	2	none				1000	10	85.1	74.5	23.3
3	670	4	rodmill	1000	35		1070	12	46.6	58.0	16.1
4	671	3	rodmill	890	34		890	13	63.0	58.9	16.5
5	663	4	rodmill	434	51		434	17.7	82.4	52.3	9.0
6	663 715	5	rodmill	451	39	35	607	26.9	82.7	80.0	11.6
7	663 715	5	rodmill	446	41	27	566	15.3	82.6	63.0	11.6
8	715 748	5	rodmill	449	50	170	1212	50.0	80.8	76.3	17.7
9	703	6	rodmill	356	52		356	20.0	80.8	74.2	12.0
10	706	4	rodmill	313	51		313	13.8	74.0	95.8	42.1
11	706	5	rodmill	326	37	31	427	15.1	93.0	87.5	42.1
12	749	7	screened through 16^m				966	26	68.5	86.0	17.5
13	749	7	screened through 16^m				949	44.2	55.4	91.2	17.5

[a] Laboratory Sample Nos.
637, 670, 671 —⅛ in. pegmatite-type waste from all sections (wet and dry) of the Chas. Bradley Mining Co. mica recovery plant in Jackson County, N. C.
649 —⅛ in. alaskite-type waste from the wet section, (4 trommels and 3 sets of rolls), of the Fred Deneen Mica Co. plant in Yancey County, N. C.
650 10-mesh alaskite-type waste from the dry section (1 set of rolls and 1 trommel screen), of the Fred Deneen Mica Co. plant in Yancey County, N. C.
663, 715, 748 —⅛ in. alaskite-type waste from the entire plant of the Southern Mica Co., Yancey County, N. C.
703 Unstained granitic-type ore taken from below surface zone, Hart County, Ga.
706 —⅛ in. rougher mica (pegmatite type) concentrate from the wet section of the Chas. Bradley Mining Co. plant in Jackson County, N. C.
749 —⅛ in. granitic-type waste from the entire mica recovery plant of F. B. Hendricks in Cleveland County, N. C.

[b] See Fig. 2.

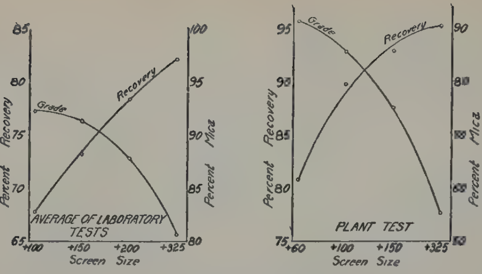


Fig. 3—Grades and recoveries obtained by screening spiral concentrates.

such as to reduce the size of gangue particle to about 16-mesh. There is some evidence that this mesh size can be extended to 10-mesh under special conditions. The size of mica flakes should not exceed about 1/3 in., since large flakes tend to catch on the edge of the ports. In practice any clean, coarse mica would normally be scalped off ahead of the spiral.

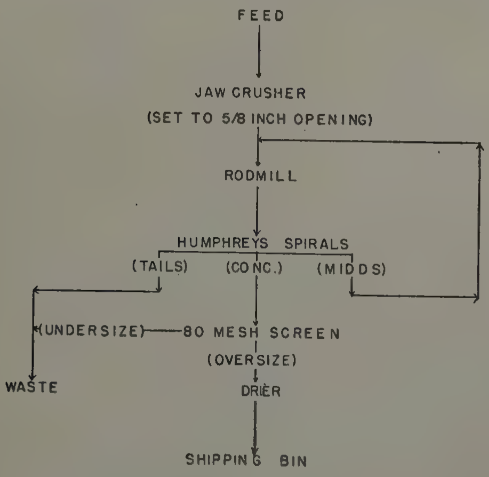
Desliming of the feed ahead of the spiral is not essential but may be desirable in order to facilitate the setting of ports and wash water. Most of the slime accompanies the mica and must be removed at some point in the process. If desliming follows the spiral, slime and water may be removed at the same time on a fine mesh dewatering screen.

Rod milling as a means of feed preparation was chosen as the most efficient means of grinding to 16-mesh without producing excessive slimes. The rod action appears to be very effective in delaminating the mica, as shown by the light weight, 19.6 lb per cu ft, of +100-mesh spiral concentrate, determined by Scott volumeter (concentrate from Test 9).

Following adequate feed preparation, the next important variable tested was feed density, which was found to be closely related to tonnage rate. Initial tests seemed to show that a low density feed, ranging from 10 to 20 pct solids gave the best results. Later laboratory tests showed that density as high as 50 pct solids was advantageous, see Tests 5, 6, 7 and 8 in Table I. The first plant test showed that feed density as high as 50 pct solids could be employed, providing the tonnage rate was sufficient to give a reasonable flow over the spiral, and the mesh size of the feed was sufficiently fine. Because of the large capacity of the standard Humphreys spiral, it has been impossible in the laboratory with present equipment to check the higher densities with their necessary high feed rates.

From discussion with engineers of the Humphreys Investment Co., it was learned that the capacity of a spiral can best be expressed in gallons per minute. This capacity will vary with the material being treated but will be fairly constant for any particular feed. Capacity figures vary from 10 to as high as 35 gpm. Because of the bulky nature of this mica feed, it was evident that the capacity would be in the lower part of this range. If 10 gpm may be considered a desirable feed rate to one spiral, the importance of using a feed density as high as possible is very apparent. At 50 pct solids this would give 1.85 tons per hr, while at 20 pct solids, feed would be only 0.58 tons per hr. At the time of this writing, feed rates have not been sufficiently steady at the first plant installation to make accurate testing possible, but from the results so far obtained, it appears that a rate of about 1½ tons per hr per spiral will be possible at pulp density approaching 50 pct solids.

Fig. 4—Simplified flowsheet for a small mica recovery plant.



Middlings Recirculation

The subject of recirculating of middlings has been carefully studied, and quite definite conclusions have been reached. With all types of mica ore tested (see Table I for description of samples) there has proved to be an advantage in returning the middlings either to the spiral or, preferably, to the grinding unit. A high proportion of the spiral middlings is composed of "booky" particles of mica, "platey" particles of gangue and, in some cases, particles of semi-kaolinized feldspar. An additional pass through the grinding circuit is necessary to further delaminate the mica particles and reduce the particle size of the other constituents. The benefits of returning the middlings may be seen from Tests 5, 6 and 10, 11.

Screening of Concentrates

The mica concentrate from spiral operation invariably contains some fine sand and usually some clay slimes. Since this mica concentrate is very dilute (under 10 pct solids), some form of dewatering screen must be employed. The choice of mesh size for this screen is determined by the compromise between recovery and concentrate grade. Curves in Fig. 3 show the effect on recovery and grade of screening these mica concentrates at various mesh sizes.

Summary

As pointed out earlier, the initial work was focused on finding a method of treating the rejects from conventional washing plants. It soon became apparent that for simplicity and low costs, the ideal installation would eliminate the rolls and screens by taking the feed directly to the rodmill, or in the case of pegmatites, follow the jaw crusher with a rodmill as shown in Fig. 4. Tests on crude ore treated in this way showed good recovery of acceptably clean mica.

In conclusion, it is believed that the process described has been sufficiently proved to justify its installation in the field. As an addition to existing plants it will recover about three quarters of the mica now being wasted, see Fig. 1. If installed as in Fig. 4, the process will successfully replace existing methods, giving a simpler, cheaper and more efficient means of recovery.

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A New Surface Measurement Tool for Mineral Engineers

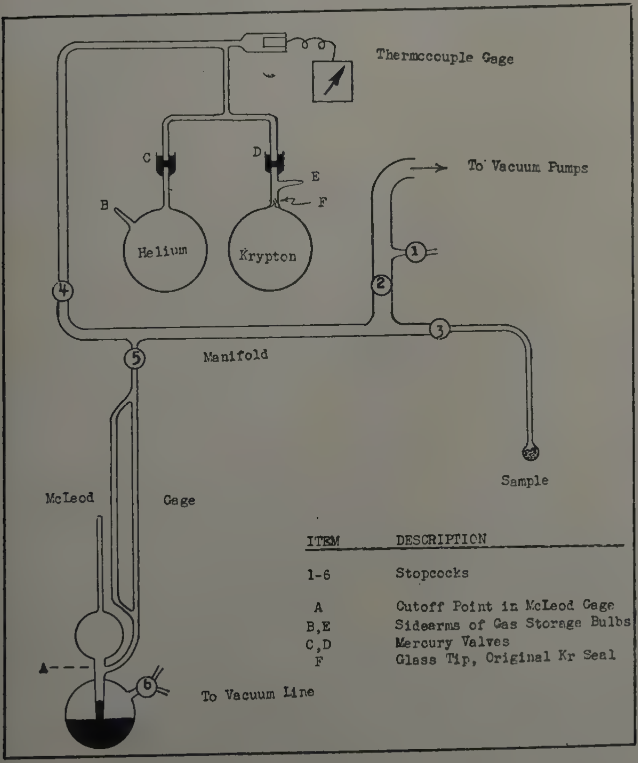
by F. W. Bloecher, Jr.

DETERMINATION of the surface area of finely divided minerals is of interest to mineral engineers engaged in flotation, comminution, and dewatering studies. In the industrial minerals field, many materials are classified as to fineness according to their specific surface (surface area per unit weight). A method of surface area determination has been tested and used in the Richards Mineral Engineering Laboratories at M.I.T. It involves low temperature krypton adsorption measurements using the technique described by Beebe.[1]

During the past fifteen years a number of new surface measurement techniques have been developed. Those that are broadly applicable to materials dealt with by mineral engineers include the permeability method[2-5] in which a gas or liquid is passed through the sample, and the low temperature gas adsorption methods first described by Brunauer, Emmett, and Teller.[6-10] Efforts to make surface determinations from size distribution curves have long been attempted[9,11,12] but because of the uncertainty of sub-sieve size distribution and inaccuracies in screen analyses, the method is at best only a rough approximation.

The most widely used gas adsorption method utilizes nitrogen gas.[6-8] Nitrogen adsorption, however, is not particularly suited as a tool for most mineral engineers because the method is not applicable for surface determinations on coarse materials, that is, materials coarser than approximately 150 to 200-mesh. Other gases have been used in mineral engineering studies, particularly ethane.[9, 10] The use of ethane gas, however, involves a technique that is too sensitive for most mineral engineering work. The permeability method of surface area determination has the disadvantage of not being sensitive enough to detect the surface represented by minute cracks, the so-called internal surface of a particle.

Fig. 1—Krypton surface measurement apparatus.



ITEM	DESCRIPTION
1-6	Stopcocks
A	Cutoff Point in McLeod Gage
B,E	Sidearms of Gas Storage Bulbs
C,D	Mercury Valves
F	Glass Tip, Original Kr Seal

The low temperature krypton adsorption method described by Beebe[1] utilizes apparatus and techniques that are essentially the same as those used in the ethane adsorption method.[9, 10] The surface area is calculated by the B.E.T. method.[6-10] The krypton adsorption technique is particularly suited to the needs of the mineral engineer because:

(1) The method can be used for making surface area measurements of rather coarse materials (35 to 48-mesh), as well as extremely fine materials, such as clays or colloidal slimes with specific surface that might be as high as 20 to 30 m^2 per g.

(2) The use of krypton gas offers the advantage of low saturation pressures (2-3 mm) at the conveniently obtained liquid nitrogen temperature. Therefore the krypton vacuum system need not be designed to measure or withstand internal gas pressures above one atmosphere as is the case with the nitrogen gas method.

(3) The krypton technique offers the safety feature of employing liquid nitrogen, as contrasted to the ethane technique in which liquid oxygen is required. Furthermore, ethane vapor pressure at liquid oxygen temperatures is low (about 10 microns), and a high sensitivity McLeod gage is required for pressure readings.

(4) The measurement technique with krypton gas is much simpler and faster, and the apparatus is less elaborate than that required with either nitrogen or ethane gases.

Apparatus

A krypton surface measurement apparatus like that used at M.I.T. is shown in Fig. 1. It is constructed of pyrex glass. A vacuum is maintained by two mercury diffusion pumps in series backed by a mechanical vacuum pump.[13] The vacuum thermocouple gage is used for checking the vacuum within the system. This type of gage is quite sensitive down to a pressure of 1 micron of mercury,[14] but may be used to indicate pressures much lower than that

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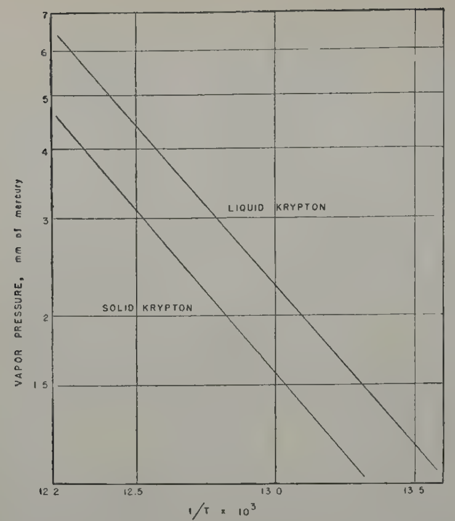

Fig. 2—Variation of krypton vapor pressure with temperature: log p vs. 1/T.

figure. If the needle on the thermocouple gage does not waver when the system is isolated from the vacuum pumps by turning stopcock 2, then the vacuum is near the required level. Such a technique, therefore, eliminates the use of more sensitive and delicate gages.

The stopcocks are a high-vacuum type with hollow plugs that are drawn into their shells by the vacuum within the system. Although high-vacuum stopcocks are more expensive than the standard type, their use will reduce, if not completely eliminate, time spent in searching for slight leaks. Stopcock 6 is a standard three-way type used for raising or lowering the mercury in the McLeod gage. One arm is connected to a water aspirator or laboratory vacuum line. Stopcock 1 is used for admitting air to the system and for evacuating the gas storage bulbs (via rubber tubing to sidearms *B* or *E*) prior to filling with helium or admitting krypton.

The mercury valves, *C* and *D*, utilize the surface tension of mercury to prevent the passage of gas out of the storage bulbs. They consist of two fritted glass faces, one attached to the storage bulb and the other to the system, both immersed in a pool of mercury. The surface tension of the mercury prevents it from passing through the frit and also prevents the gas from escaping. When the two fritted faces are placed in contact, the mercury between them is displaced, and the gas passes slowly through the frit into the system. Small quantities of gas thus can be admitted to the system at a very slow rate.[15]

Krypton gas can be purchased from the Air Reduction Co. in a sealed pyrex storage bulb. The lower half of the mercury valve and a side arm *E*, must be attached to the neck of the bulb. A small iron nut is placed in the side arm. The arm is connected to the system at stopcock 1 by pressure tubing, and all the air is pumped out of the neck of the krypton bulb. The side arm is then sealed off by heating with a torch. The small tip *F* sealing the krypton in the bulb is broken by manipulating the iron nut with a hand magnet.

The helium is ordinary tank grade (99.8 pct pure) and is admitted to the bulb through pressure tubing connected to the tank. A glass tee placed in the tube leading from the tank will provide means for a hose connection to the system at stopcock 1, thereby enabling the air to be evacuated from the bulb and tube prior to admitting helium. The helium is passed through a trap immersed in liquid nitrogen in order to remove any water vapor or other condensable gases which it might contain. Helium pressure in the storage bulb must not be allowed to exceed one atmosphere, so that the side arm *B* can be sealed off by heating with a torch.

The low-sensitivity McLeod gage has been calibrated linearly[16] so that pressures from 0.1 mm to 2.7 mm can be measured with it to three significant figures. The volume of the bulb of the gage (above cut-off *A*) was determined quite accurately by filling with mercury and weighing before it was placed in the system. With its volume known (41.78 cc), the gage can be used as a buret for determining volumes of other parts of the system and for measuring quantities of krypton gas used in the adsorption measurements. For this purpose the fundamental gas relationship, $P_1V_1 = P_2V_2$ is used.[9] The manifold volume of the system illustrated (that volume bounded by stopcocks 2, 3, and 4, and cut-off *A*) was 33.61 cc. The reproducibility of a series of such volume determinations was found to be of

Table I. Krypton Adsorption Data on 1.0302 G of Specular Hematite, Cone 5 Infrasizer Fraction

	1	2		3	4	5	6	7	8	9	10
		Room Temp.									
Step	Description	°C	°K	Krypton Pressure, P_1[a]	Amount of Kr, cc-mm, P_1V_1[b]	Adsorption Pressure, p	Amount Kr in V_2, cc-mm, pV_2[b]	Kr Adsorbed, cc-mm, $P_1V_1 - pV_2$	Kr Adsorbed, cc at STP, V	$\frac{p}{V(P_o-p)}$	$\frac{p}{P_o}$
1	Kr in	24.4	297.5	2.14	161.33	1.37	106.52	54.81	0.06620	11.2[e]	0.425[e]
2	Kr out				—103.28[c]						
3		24.4	297.5		58.05	0.420	32.66	25.39	0.03066	4.89	0.130
4	Kr out				—14.12[b]						
5		24.5	297.6		43.93	0.307	23.87	20.06	0.02422	4.35	0.0953
6	Kr out				—23.14[c]						
7	Net Kr				20.79						
8	Kr in	24.6		1.38	104.04						
9	Total Kr	24.6	297.7		124.83	1.04	80.86	43.97	0.05307	8.99	0.323
10	Kr out				—34.95[d]						
11		24.6	297.7		89.88	0.717	55.75	34.13	0.04119	6.95	0.223
12	Kr out				—24.10[d]						
13		24.5	297.6		65.78	0.494	38.41	27.37	0.03304	5.48	0.153

[a] Pressures are in mm of mercury.

[b] $V_1 = 75.39$ cc (except in steps 4, 10 and 12 where $V_1 = 33.61$ cc) $V_2 = 77.75$ cc.

[c] Kr pumped out of volume V_1; i.e., in step 2, 75.39 x 1.37, or 103.28 cc-mm of Kr removed.

[d] Kr pumped out of manifold volume only; i.e., in step 4, 33.61 x 0.420, or 14.12 cc-mm of Kr removed, (33.61 cc = manifold volume).

[e] P_o = saturation pressure of super-cooled liquid Kr, 3.22 mm. Observed pressure of solid Kr was 2.25 mm (see fig. 3).

the order of ±1 pct. Helium gas was always used in making volume measurements.

Krypton Adsorption Data

The technique of obtaining krypton adsorption data is similar to that used by others with the ethane apparatus.[9, 10] A representative sample is weighed out in a suitable pyrex bulb and sealed to the system with a glass blowing torch. A total sample surface of at least 1000 cm² is required to insure good experimental reproducibility. If the sample contains adsorbed water, as is the case with certain clays, it should be heated at about 100°C until it reaches a constant dry weight before it is sealed to the system.

After the sample is attached to the system, the air is pumped out of the sample chamber. The initial evacuation of air should be started with extreme caution by carefully opening stopcock 3. If this stopcock is opened too suddenly, the sample will be sucked into the system. The sample is heated to about 100° to 200°C with a small electric furnace and evacuated for several hours to remove all adsorbed gases. The thermocouple gage can be used to measure the completeness of this "outgassing" of the sample surface. Three to four hours evacuation will usually suffice.

After evacuation the heat source is removed and the sample is allowed to reach room temperature. The combined manifold, gage and dead space volume is determined using the McLeod gage as a buret (the dead space is the volume surrounding the sample beyond stopcock 3). The gage and manifold volumes, of course, will be constant for a given vacuum system, but the dead space volume will change every time a new sample is sealed to the system.

Before starting krypton adsorption measurements, the sample must be cooled to liquid nitrogen temperature. To speed up this cooling, helium gas (approximately 1 cm pressure) is admitted to the dead space to act as a heat transfer medium. Ten to fifteen minutes were found to be sufficient to insure that the sample temperature had reached the desired level. With the sample bulb remaining in liquid nitrogen, the helium is then completely pumped out. No helium is adsorbed by the sample at liquid nitrogen temperature.

Some krypton is admitted to the system through the mercury valve *D*, stopcock 3 being closed. It occupies a volume V_1, consisting of the gage plus manifold volumes, and exerts a pressure P_1, as noted in line 1 of Table I, a collection of adsorption data on specular hematite. There are a total of P_1V_1 cc-mm of krypton occupying the volume V_1, as recorded in col. 4 of Table I. By opening stopcock 3 the krypton is allowed to expand into the dead space, the sample tube still being immersed in liquid nitrogen. After allowing about 5 to 10 minutes for adsorption equilibrium to set in, the adsorption pressure p is measured with the McLeod gage and recorded (col. 5). The total amount of gaseous krypton remaining in volume V_2 (gage plus manifold plus dead space) is then pV_2 cc-mm, recorded in col. 6. The difference between the values of col. 4 and 6, which is recorded in col. 7, is the cc-mm of krypton adsorbed by the sample at the adsorption pressure p. The factor, $\frac{273.1}{°K} \times \frac{1}{760}$, is used to convert the cc-mm of col. 7 to the cc at standard temperature and pressure (STP) recorded in col. 8. The pressure, P_o, used in calculating the values in col. 9 and 10 is the saturation pressure of liquid krypton at the temperature of absorption and is not measured until after a series of adsorption measurements have been completed, i.e., until after the data of line 13 has been collected.

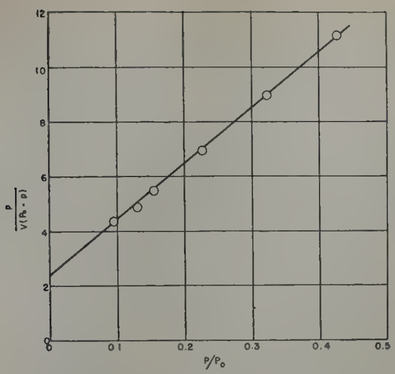

Fig. 3—B.E.T. plot of krypton adsorption on specular hematite.
Slope = 20.5
Intercept = 2.37
Area = 2230 cm²
per g

After measuring the adsorption pressure p of line 1, stopcock 3 is closed, and the krypton remaining in the volume V_1 (gage plus manifold) is completely pumped out. That is, pV_1 cc-mm of krypton are removed from the system, as shown in line 2. The remaining krypton is allowed to redistribute itself throughout the volume V_2 and the adsorption data of line 3 is collected as was that of line 1.

Stopcock 3 is closed again to allow for the krypton removal step of line 4. However, in this gas removal step, the krypton is pumped out of only the manifold volume, the mercury being allowed to remain above the cut-off in the McLeod gage, thereby retaining the krypton in the gage volume as well as that in the dead space.

Then another set of adsorption readings is obtained and recorded as per line 5.

The rest of the data in Table I is obtained in a similar manner, by addition of more krypton (line 8) followed by alternate expansion and evacuation.

After the adsorption data is collected, krypton gas is admitted to the system until it is visibly condensed on the walls of the sample tube. The vapor pressure of this condensed krypton is measured allowing sufficient time for pressure equilibrium to set in. According to Beebe,[1] it is the vapor pressure of supercooled liquid krypton which should be used in the

Table II. Some Representative Surface Area Measurements

Material	Weight, g	Description	Surface Area, cm² per gram[a]
Monazite conc., a	1.314	48/100-mesh	2,140
Monazite conc., b	2.423	48/100-mesh	1,980
Monazite	0.1582	1-hr grind	13,550
Monazite	0.1278	2-hr grind	23,700
Monazite	0.1301	4-hr grind	42,400
Monazite	0.1060	8-hr grind	69,000
Monazite	0.1044	18-hr grind	135,000
Smithsonite	1.671	100/270-mesh	5,440
Smithsonite	1.671	100/270-mesh	5,300
Smithsonite	1.671	100/270-mesh	5,300
Smithsonite	1.671	100/270-mesh	5,350
Quartz	1.257	—200-mesh, deslimed at about 400-mesh	897
Sphalerite	19.14	65/100-mesh	226
Sphalerite	19.14	65/100-mesh	225
Ilmenite-Leucoxene conc.	2.294	48/100-mesh	42,100
Ilmenite-Leucoxene conc.	0.2621	48/100-mesh heated in 1:1HCL for 1 hr, water washed and dried.	57,700
Specular Hematite	1.030	Cone 4 infrasizer fraction	2,230

[a] The area per krypton molecule was taken as 19.5 sq A.

B.E.T. equation, since the krypton is adsorbed as a liquid rather than a solid layer. Therefore, the measured vapor pressure must be converted to the liquid krypton pressure using the chart of Fig. 2, drawn from the data of Meihuizen and coworkers.[17, 18] The adjusted P_o value is then used in calculating the figures in cols. 9 and 10 of Table I.

Interpretation of Data

The B.E.T. equation is usually written in the form:[6, 7]

$$\frac{p}{V(P_o - p)} = \frac{1}{V_m C} + \frac{(C\text{-}1)\,p}{V_m C\ P_o} \qquad [1]$$

where V is the volume of krypton (cc at STP) adsorbed at a pressure p, P_o is the vapor pressure of super-cooled liquid krypton at the temperature of adsorption, V_m is the amount of gas (cc at STP) required to form a monolayer on the sample, and C is a constant related to the heat of adsorption of krypton.

By making a plot of the experimental values of $\frac{p}{V(P_o - p)}$ as the ordinate, against p/P_o as the abscissa, a straight line results over a p/P_o range of about 0.05 to 0.35. The slope of that line will be equal to $\frac{(C\text{-}1)}{V_m C}$, and the intercept equal to $\frac{1}{V_m C}$. Solving the two equations simultaneously results in the simple equation for V_m of:

$$V_m = \frac{1}{\text{slope} + \text{intercept}} \qquad [2]$$

Fig. 3 is a B.E.T. plot of the data in cols. 9 and 10 of Table I. The slope of the line is 20.5, and its intercept 2.37. Substitution of those figures in eq 2 gives 0.04373 cc of krypton as the amount required to form a monolayer of krypton on the surface of the hematite sample.

The specific surface of the sample is calculated using the equation:

$$S = \frac{V_m \times 6.023 \times 10^{23} \times 10^{-16} \times d}{22400 \times \text{wt of sample}} \qquad [3]$$

where 6.023×10^{23} is the number of molecules in 22,400 cc of krypton gas at STP (Avogadro's number), d is the cross-sectional area of a krypton molecule in square angstroms, and 10^{-16} is the factor converting square angstroms to cm^2. According to Beebe's data,[1] the area per krypton molecule is approximately 19.5 sq Å. That figure has been checked closely by making a surface area determination on a sample of zinc oxide previously measured by the nitrogen adsorption technique. Beebe's figure of 19.5 sq Å has been used for the work reported herein.

Inserting the weight of the hematite sample and the calculated value of V_m in eq 3 gives a specific surface of 2230 cm^2 per g for that material.

Typical Surface Measurements

Table II contains a representative list of materials on which surface area determinations have been made with the krypton apparatus. A glance at the table will show that the method is quite well suited for use in the mineral engineering laboratory. The coarsest material on the list, from a surface area standpoint, had a specific surface of only 225 cm^2 per g, while the finest material measured had an area of 135,000 cm^2, or 13.5 m^2, per g. The series of runs on the 1.671 g sample of smithsonite shows that the reproducibility of experimental data is well within + or − 2 pct. The two runs on the coarse sphalerite show that the experimental reproducibility is excellent even when dealing with rather coarse materials.

Acknowledgment

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Application of Cyclone Thickeners To Preparation Plant Water Circuits

by G. H. Kennedy and H. E. Criner

CYCLONE thickeners are "hydraulic centrifuges" designed either to produce a maximum practical separation between the water and solids of a feed slurry or to classify the slurry solids according to particle settling rate. The possibilities of utilizing the thickener to "close" plant water circuits have been investigated.

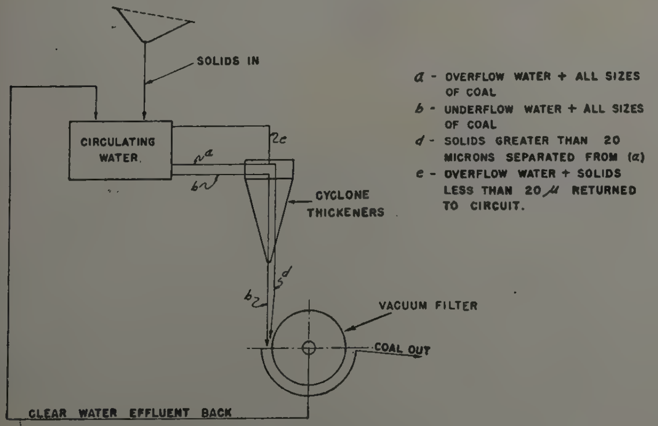
Since the thickener becomes an integral part of the plant water circuit when installed, the basic means by which the solids content of all circulating water systems must be controlled is reviewed.

All coal not separated from the circulating water by means of dewatering screens or settling devices must be bled from the plant at the same rate as they are being fed or created by degradation. If this were not true, the circulated volume would finally be completely filled with solids.

If the rate of bleed is low, the solids concentration in the bleed and circulating water generally will be high. To keep a reasonable circulating water concentration, large amounts of fresh water must be supplied and the bleed rate must be high. These conditions are usually undesirable because of the difficulty in securing fresh water, the loss of large quantities of coal, and the pollution of nearby streams.

Thickeners are the primary stage in the separation of the water contained in the bleed from the coal. Since a thickener, when operating with reasonable efficiency, usually cannot be expected to deliver an underflow containing less than 50 pct water, secondary dewatering devices such as a slurry screen or vacuum filter must be used to separate additional water from the bleed. The surface area required by the filter or screen will be nearly proportional to the total water which must be handled, and therefore the thickener becomes necessary to hold down the original cost. The water separated from the coal is then returned to the plant circuit to maintain the water balance and thus becomes the source of fresh water.

Fig. 1—Elements of closed-circuit water system.


SOLIDS IN
CIRCULATING WATER
a
b
c
CYCLONE THICKENERS
d
b
VACUUM FILTER
COAL OUT
CLEAR WATER EFFLUENT BACK
a - OVERFLOW WATER + ALL SIZES OF COAL
b - UNDERFLOW WATER + ALL SIZES OF COAL
d - SOLIDS GREATER THAN 20 MICRONS SEPARATED FROM (a)
e - OVERFLOW WATER + SOLIDS LESS THAN 20 µ RETURNED TO CIRCUIT.

It is not necessary to return clear water to the plant to maintain control of the circulating water concentration. It is only necessary that the bleed dewatering devices remove coal in all sizes at a rate equal to that at which these sizes are entering the circulating water. However, practically speaking, in order to maintain low solids concentration, it is usually necessary that the thickener have good recovery characteristics on bituminous coals in those sizes down to 400-mesh.

The thickener can remove the smaller sizes from the closed-circuit system at the "solids in" rate and prevent their "build up" in the circulating water as shown in Fig. 1. With the portion of the feed water going directly to the underflow, comes the very fine solids not concentrated by the cyclone. All the larger solids are removed from the feed and delivered in a concentrated condition at the underflow.

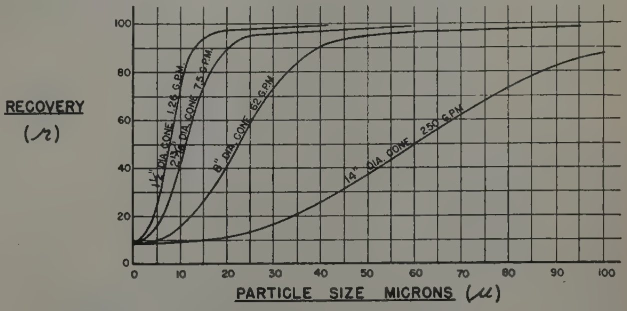


Fig. 2—Relative recovery efficiencies of various sizes of cyclone thickeners.

To apply thickeners with some reasonable assurance that operating requirements will be met, it is necessary to know the rate at which solids of all sizes must be removed from the circulating water and to know the thickener characteristics.

The required removal rate of fine solids not recovered by other dewatering devices may be estimated from the size distribution and tonnage of the plant feed with a reasonable allowance for degradation in the circulating water system. A more reliable estimate may be secured by sampling the plant bleed water during a constant load period.

G. H. KENNEDY, Preparation Engineer, is with the Rochester and Pittsburgh Coal Co., Indiana, Pa. H. E. CRINER, Development Engineer, is associated with Heyl and Patterson, Inc., Pittsburgh, Pa.

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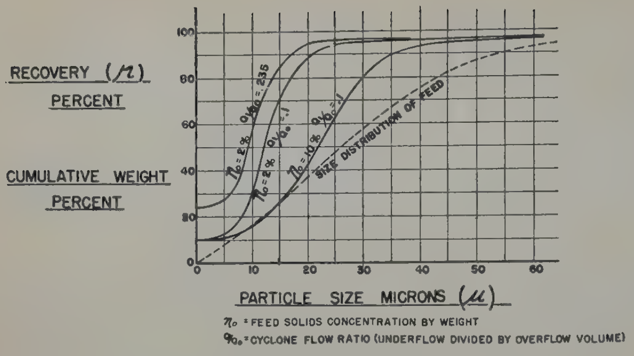

Fig. 3—Influence of feed concentration and flow ratio on recovery of 2 13/16-in. diam cyclone thickener.

The thickener characteristics will depend upon the design of the unit and the operating conditions. Fig. 2 shows the influence of one design factor, i.e., size. Fig. 3 shows the effect of feed concentration and flow ratio. It is to be noted that increasing cyclone size, feed concentration, and decreasing flow ratio will all serve to lower thickener efficiency.

If the above factors are known, a good estimate of the resultant circulating water and filter feed concentration and size distribution may be calculated.

A cyclone thickener, vacuum filter combination was installed in September 1949, at the Kent Plant of the Rochester and Pittsburgh Coal Co. This paper contains operating data obtained at this installation.

The preparation plant is a combination of sand flotation system washing 5x¼ in. coal and concentrating tables washing ¼ x 0 coal at a rate of 140 tph. The washed ¼ x 0 is collected in two settling tanks having a total volume of 65,000 gal connected about 2 ft below the water level. The water circuit of the coarse and fine coal plants are connected.

The effluent from the Centrifugal and Mechanical Industries centrifuges is returned to the settling tanks. The plant operates three shifts per day.

The ¼ x 0 coal is centrifugally and thermally dried; the ¾ x ¼ is thermally dried.

Initially the water clarification device was a solid bowl centrifuge. The feed consisted of a mixture of the effluent from the CMI's and circulating water. The feed rate was 400 gpm at 30 pct solids. The plant bleed was a portion of the effluent. The bleed rate was 100 gpm at 12 pct solids which were almost all smaller than 37 microns. The circulating water averaged about 25 pct; the cake averaged 30 pct moisture.

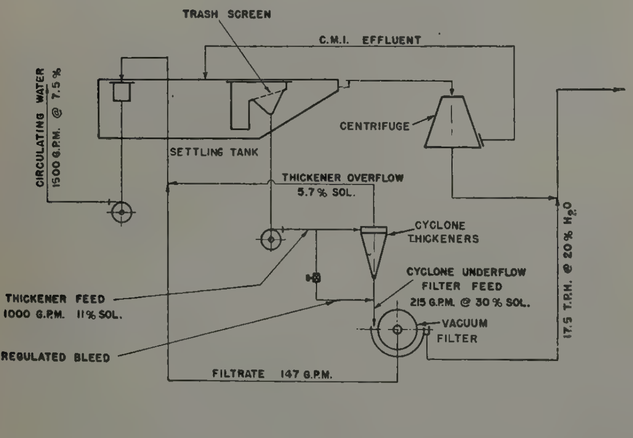
TRASH SCREEN
C.M.I. EFFLUENT
CIRCULATING WATER 1500 G.P.M. @ 7.5 %
CENTRIFUGE
SETTLING TANK
THICKENER OVERFLOW 5.7 % SOL.
CYCLONE THICKENERS
CYCLONE UNDERFLOW FILTER FEED 215 G.P.M. @ 30 % SOL.
THICKENER FEED 1000 G.P.M. 11 % SOL.
VACUUM FILTER
REGULATED BLEED
FILTRATE 147 G.P.M.
17.5 T.P.H. @ 20 % H₂O

Fig. 4—Kent water circuit.

A detailed study, December 1947, showed that 17.5 tph of fines smaller than ½ mm must be removed in order to maintain an equilibrium of 7.5 pct solids in the circulating water and operate in closed circuit. It was decided that the plant should be so designed that it would be possible to dispose of an effluent containing no appreciable amount of coal.

Based on this study it was decided to install Heyl and Patterson cyclone thickeners and an Oliver disc type vacuum filter.

Fig. 4 is a flow diagram of the present circulating system. The cyclone feed passes through a ⅛-in. round hole, self-cleaning trash screen, which removes wood chips, waste and other foreign matter which would plug the cyclones. The feed pump supplies 1000 gpm at 40 psi to 4 banks of 3-in. diam cyclones, each containing 22 cyclones. The feed line to each bank is valved so that any number of banks may be used. The underflow is pumped to a 8 ft 6 in. x 10 disk vacuum filter. The overflow from the thickeners joins the filtrate and is returned to the settling tanks at the point of the circulating water take off. The filter cake is discharged by way of a belt conveyor to the same conveyor that handles the centrifuged coal to a 25-ton surge bin ahead of two flash driers. The coal is conveyed from the bins to the flash driers by screw conveyors.

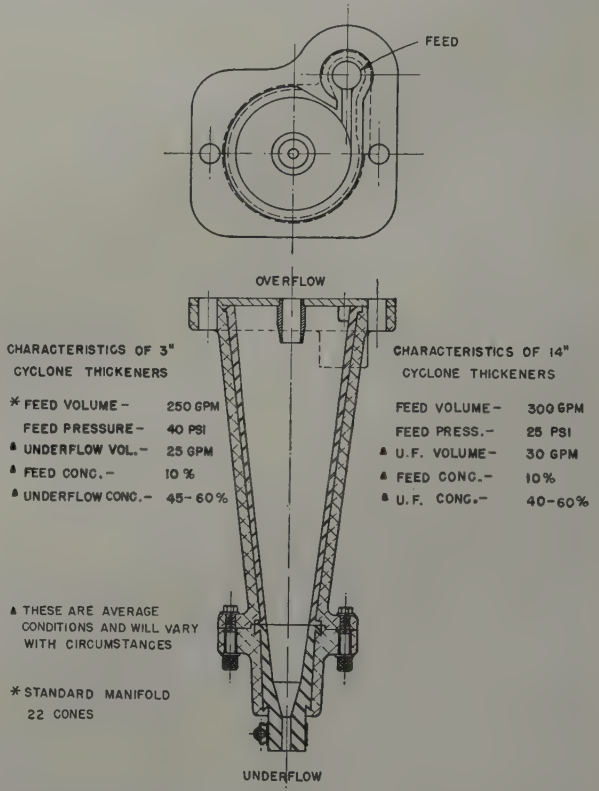
FEED
OVERFLOW
CHARACTERISTICS OF 3" CYCLONE THICKENERS
* FEED VOLUME - 250 GPM
FEED PRESSURE - 40 PSI
▲ UNDERFLOW VOL. - 25 GPM
▲ FEED CONC. - 10 %
▲ UNDERFLOW CONC. - 45-60 %
CHARACTERISTICS OF 14" CYCLONE THICKENERS
FEED VOLUME - 300 GPM
FEED PRESS. - 25 PSI
▲ U.F. VOLUME - 30 GPM
▲ FEED CONC. - 10 %
▲ U.F. CONC. - 40-60 %
▲ THESE ARE AVERAGE CONDITIONS AND WILL VARY WITH CIRCUMSTANCES
* STANDARD MANIFOLD 22 CONES
UNDERFLOW

Fig. 5—Rubber-lined cyclone thickener, 3-in. diam aluminum body.

Fig. 5 is a cross section through the Heyl and Patterson 3-in. diam cyclone thickener. The casing is made in two parts, the main body and the underflow orifice holder. The material is die cast aluminum. The rubber lining is mounted in the case so that it can be replaced easily. The underflow orifice piece is rubber, and adjustment for wear is made by a simple hose clamp.

The following plant conditions have prevented complete operation of the cyclones:

(1) The bins ahead of the flash drier have a

tendency to plug when fed with much more than 12 tph of filter cake, so that it has never been possible to produce as much filter cake as required to bring the circulating water down to the concentration for which the system is designed.

(2) The overflow from the filter is returned to the sump ahead of the trash screen, resulting in recirculation of thickened material and excessive wear of the cyclones.

(3) The rate of fresh water input in the form of gland-sealing water has always been in excess of the water evaporated and lost on the coal surface so that there has always been a plant bleed.

For a special 5-hr test period the water circuit was modified to permit closed-circuit operation. During this time the following conditions were found to exist: circulating water concentration, 12.7 pct; cyclone thickener feed concentration, 18.6 pct; cyclone thickener underflow concentration, 55.6 pct; cyclone thickener overflow concentration, 11.5 pct; vacuum filter feed concentration, 37.2 pct; vacuum filter cake moisture (wet basis), 21.1 pct; and vacuum filter cake rate, 26 tph.

Fig. 6 shows the particle size distribution of the cyclone thickener products.

The increased filter cake rate can be attributed to an increased plant load of ¼ x 0 coal and a corresponding increase in —½ mm solids in the circulating water. When recalculated for this new condition, the equilibrium circulating water concentration would be 13.5 pct. Such close correspondence between calculated and actual results is possible when an understanding of the cyclone thickener characteristics is coupled with a detailed knowledge of the plant circuit and a knowledge of the rate at which solids must be removed from the system.

However, without the thickeners operating, the vacuum filter alone permitted the circulating water concentration to increase from about 13 pct to 25 pct within a three-shift period. Furthermore, when neither the vacuum filter nor the cyclone thickeners were in operation, the concentration of the water rapidly increased to 42 pct and necessitated a plant shutdown. The beneficial effect of the cyclone thickeners on the system is conclusive and the following are several specific improvements: (1) Positive control of the circulating water concentration has been made possible, (2) lower circulating water concentration has resulted in improved table performance, (3) fine coal lost to the silt ponds has been reduced, (4) abrasion rate in pumps and piping has been reduced, (5) pipe line plugging has been eliminated substantially, and (6) storage of concentrated slurry is at a minimum.

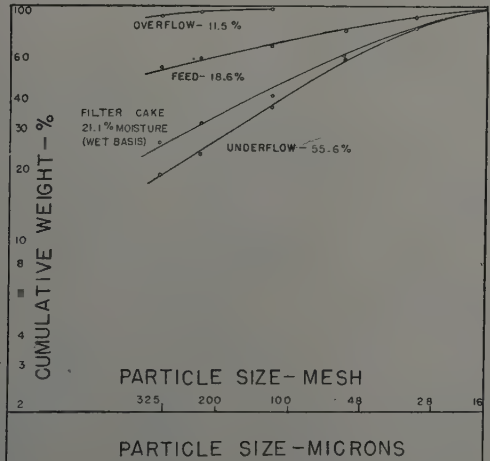

Fig. 6—Size distribution curves for 3-in. diam cyclone thickener products. Rochester and Pittsburgh Coal Co., Kent No. 1 and No. 2.

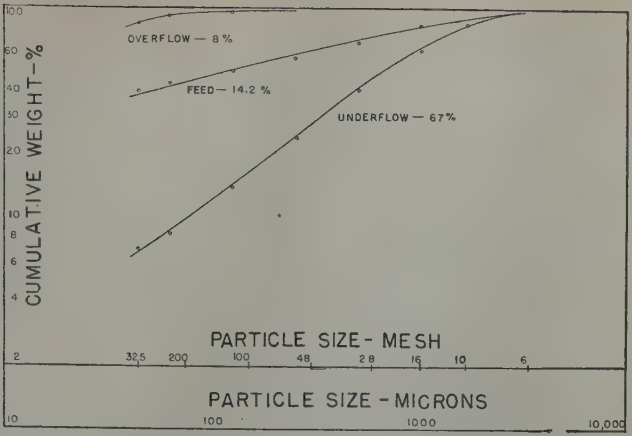

Fig. 7—Size distribution curves for 14-in. diam cyclone thickener products. Carpentertown Coal and Coke Co.

In addition to these operational improvements it is important to note the small space requirement for a cyclone thickener installation. Also, since it is within the plant structure, it is often possible to install it adjacent to the vacuum filter, permitting easy control of these interdependent operations.

Maintenance for the 3-in. cyclone thickeners consists chiefly of replacement of rubber parts subject to the abrasive action of the coal slurry. Field experience indicates that the rubber lining of the cone body will be serviceable for 1000 to 1500 hr depending upon the severity of the application. The rubber underflow orifice has a corresponding life of about 300 to 400 hr. Both parts are designed for rapid removal and replacement.

Another closed-circuit system using four 14-in. diam cyclone thickeners has been installed at Carpentertown Coal and Coke Co., Carpentertown, Pa. Through the courtesy of Mr. L. O. Lougee and Mr. C. Cornelius the following information is available.

Prior to the cyclone installation, fine solids were removed from the system on a 4x10-ft high speed dewatering screen having ¾-mm opening woven cloth and with a continuous plant bleed to a pair of settling ponds. The combined removal rate was estimated to be about 20 tph. Analysis of the solids to be removed revealed that only 5 pct passed 200-mesh and that 100 pct passed a 14-mesh sieve. This small percentage of very fine material made it possible to utilize the existing dewatering screen to further dewater the cyclone underflow. Thickening of the screen feed increased the screen retention to the point where closed-circuit operation was a reality. Test results showed the following operating performance: cyclone thickener feed, 14.2 pct; cyclone thickener underflow, 67 pct; cyclone thickener overflow, 8 pct; dewatering screen product moisture (wet basis), 25 pct; dewatering screen solids rate, 22 tph.

Fig. 7 shows the particle size distribution of the cyclone thickener products.

The two systems described serve to substantiate the claim that it is possible to calculate cyclone thickener capacity requirements, to estimate the performance with reasonable accuracy, and to close preparation plant water circuits in conjunction with dewatering screens or vacuum filters.

An Investigation of the Abrasiveness of Coal and Its Associated Impurities

by H. F. Yancey, M. R. Geer, and J. D. Price

COAL mine operators recognize coal as an abrasive material, because the wear of drilling, cutting, and conveying equipment is reflected as a cost item for replacement of parts. Similarly, industrial consumers of coal experience abrasive wear on all coal-handling equipment. Operators of pulverized fuel plants are doubtless most keenly aware of the abrasiveness of coal, because under the high contact pressures developed between coal and metal in pulverizers, abrasive wear is increased many fold. Moreover, experience in operating pulverized fuel plants has demonstrated that some coals are much more abrasive than others.

Hardgrove[1] stated that maintenance costs entailed by the wear of grinding elements is often a more important variable than the cost of the power required to pulverize different coals. Craig[2] also reports that one coal may cause pulverizer parts to wear several times faster than another. It is apparent, therefore, that those concerned with pulverizing coal could profitably employ a method for estimating the abrasiveness of different coals, just as they utilize standard tests for thermal value, grindability, and ash-fusion temperature to assist in selecting the most suitable and economical coal to use in a particular plant.

The objective of this investigation was to develop a test procedure that would be suitable for general use in estimating the abrasiveness of coals. However, few, if any, of the standard tests now used for evaluating the properties of coal are the product of a single investigation or the result of a single investigator's efforts. Rather, in each case, a testing procedure was devised by one investigator, used by others on a wider variety of coals, and finally refined completely as the result of the joint efforts of a number of interested people. Thus, the test procedure for estimating abrasiveness developed in the course of this work may not be refined sufficiently in its present form for general use, but it may serve as the starting point from which an acceptable test procedure can be developed. The method has been used thus far on only about a dozen coals, and there has been no opportunity to attempt a correlation between experimental results and actual plant experience. Only wider use of the procedure by other investigators and correlation with plant experience can determine to what extent the method will have to be modified to render it suitable for general application.

Test Method

Although the literature contains no record of an attempt to devise a method for estimating the abrasiveness of coal that could be used industrially, several investigators have tested properties of coal that are closely related to its abrasiveness. The abrasiveness of a material generally is considered to be related to its hardness, and hardness tests for coal have been employed by Heywood,[3] O'Neill,[4] and Mathes.[5] Also, the resistance of coal to abrasion, a property that presumably is related to the abrasiveness of coal, was measured by Heywood[3] and by Simek, Pulkrabek, and Coufalik.[6] All these investigators tested only individual pieces of coal. Since coal is a heterogeneous material having components of varying properties, tests of this type can yield results having little more than academic interest. Only a test method that utilizes a representative sample of coal can give results that are useful industrially.

The abrasion tests used for various other materials have been considered for adaptation to testing the abrasiveness of coal. The tests used for metals,[7-9] paving and flooring,[10] and rubber,[11] cannot be used because coal is not sufficiently abrasive.[4]

The present experimental work was begun before World War II and was conducted by three research fellows[12-14] working under a joint agreement between the University of Washington and the Bureau of Mines. After a great deal of preliminary work with a variety of apparatus and materials, a test procedure was developed which consisted of rotating a test disk 2½ in. diam in a steel mortar containing the coal sample. The shaft carrying the test disk at the lower end and a 100-lb load on the upper end was free to move vertically. The bed of coal in the mortar was kept fluid by low-pressure air admitted through a port near the bottom of the mortar. Measurable wear on an Armco iron disk could be obtained in this test procedure, but, despite extensive efforts to eliminate them, several major disadvantages remained in this test method. First, with most coals the amount of wear on the iron disk did not exceed a few milligrams. Second, a single type of disk was not applicable for all coals. A smooth iron disk gave satisfactory results with both bituminous and sub-bituminous coals, but hardly any wear with anthracite or coke. A disk having studs or projections gave more satisfactory abrasion losses with anthracite and coke and presented no operating difficulties with free-burning bituminous and sub-bituminous coals. It could not, however, be used with caking coals because these coals formed a

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cake on the face of the disk, preventing abrasive wear.

During the war years this problem was recessed for more important work, and when it was resumed the disadvantages of the earlier test procedure seemed so great that the method was abandoned and a fresh approach made to the problem.

Equipment and Procedure

The present abrasion apparatus is illustrated in Figs. 1 and 2. Essentially, the test consists of rotating four removable wearing blades in a charge of coal for a fixed number of revolutions and then determining the loss in weight sustained by the blades during the test. This loss in weight, expressed in milligrams, is considered the abrasiveness of the coal.

The apparatus consists of a shaft-and-arm assembly to hold the wearing blades, a mortar that holds the charge of coal, and a drill press to provide rotation.

The blades are 2 in. long by 1½ in. wide, and are made from 16-gage Armco iron. The edge of the blade next to the wall of the mortar is curved backward with respect to the direction of rotation on a ⅝-in. radius. The purpose of this curvature is to crowd or *nip* the coal particles between the blades and the wall of the mortar. A clearance of 0.20 in. is maintained both between the blades and the bottom of the mortar and between the blades and the wall of the mortar.

The blades are carried on four radial arms spaced 90° apart. These arms, shaped to fit the curved blades and provide support for them, are held in a hub on the shaft by set screws so that the length of the arms and therefore the clearance between the blades and the wall of the mortar can be adjusted.

The shaft carrying the arms is turned to semi-hemispherical shape at the lower end to fit a thrust bearing. The upper end of the shaft fits into the spindle of the drill press, where it is held by set screws.

The mortar, turned from extra-strength pipe, is 8 in. ID by 9 in. deep. A simple brass thrust bearing screwed through the bottom of the mortar supports and centers the shaft. Adjustment of this bearing maintains a 0.20-in. clearance between the bottom edge of the wearing blades and the bottom of the mortar. Four ⅜-in. square bars fastened to the underside of the mortar bottom radially at 90° intervals fit into matching grooves in the table of the drill press to center the mortar under the spindle of the drill press and prevent the mortar from rotating. A tight-fitting cover on the mortar is provided with a clearance hole for the rotating shaft.

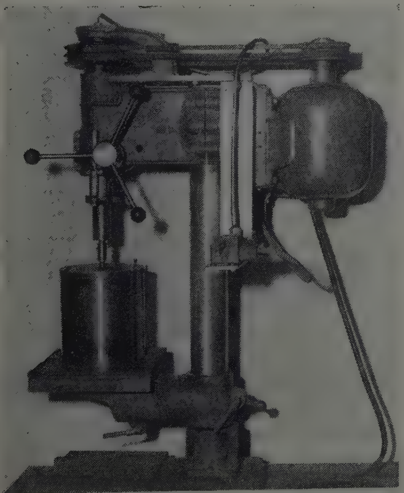

Fig. 1—Assembled abrasion apparatus.

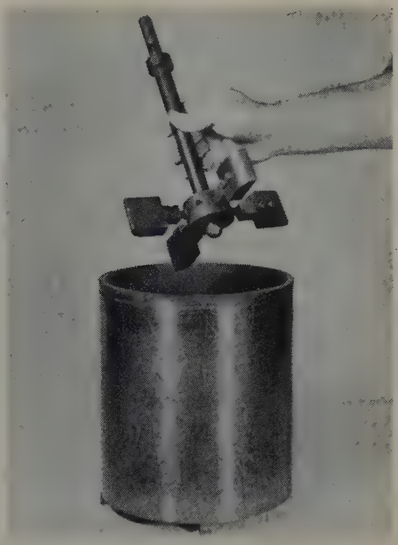

Fig. 2—Mortar, shaft-and-arm assembly, and wearing blades.

The drill press is powered with a 3-hp motor and provided with sheaves to give a spindle speed of 1500 rpm. This amount of power is not required for ordinary testing but was necessary for tests in which the clearance between the wearing blades and the wall of the mortar was reduced substantially below 0.20 in. A Veeder-Root revolution counter to shut off the motor after a predetermined number of revolutions completes the abrasion apparatus.

In making a test, the four blades are weighed on an analytical balance to the nearest milligram. They are then fastened to the supporting arms, and the shaft assembly is placed in the mortar with the lower end of the shaft engaged in the thrust bearing. A 4-kg sample of air dried 4-mesh to 0 coal, prepared by stage-crushing with smooth rolls, is poured into the mortar.

After the rotating period, the wearing blades are removed, cleaned, cooled in a desiccator, and again weighed. With many coals, the blades can be cleaned adequately by merely rubbing with a dry cloth. Some coals, however, leave a residual film on the blades that is removed by rubbing with a damp cloth, then scouring with a fiber-brush wheel, and finally rubbing again with a damp cloth. The moisture from the damp cloth is evaporated quickly by the heat of the blades.

Coals Tested

Table I identifies the 14 coal samples tested by this test procedure. Most of these are from Washington, but a few representative coals from other states were included. These samples range in rank from anthracite to sub-bituminous B and cover a wide range in ash content. Float-and-sink fractions of a number of these samples were tested separately to illustrate the relative abrasiveness of the clean coal and its associated impurities.

Not all of the coal samples were tested after the operating variables of the test method were standardized, and consequently the data in some tables cannot be compared directly with similar data in other tables.

Influence of Variables

In developing a new test procedure, the exploration of the variables inherent in the procedure constitutes the major part of the investigation. Five principal variables in the present test procedure influence the magnitude of the abrasion loss ob-

Table I. Identification of Coal Samples Tested

State	County	Mine	Bed	Size[a]
Illinois	Christian	Langley No. 9	No. 6	Screenings
Pennsylvania	Indiana	Cush Creek	Upper Kittanning	Run-of-mine
Pennsylvania	Allegheny	Montour No. 10	Pittsburgh	2x0 in.
Pennsylvania	Schuylkill	Coaldale	Mammoth (?)	Pea
Utah	Carbon	Castle Gate	D	1⅝x0 in.
Utah	Carbon	King	Hiawatha	Stoker
Washington	King	McKay	McKay	1x0 in.
Washington	King	Olson-Cumberland	Unnamed	Run-of-mine
Washington	Kittitas	Roslyn No. 3	Roslyn	Run-of-mine
Washington	Kittitas	Roslyn No. 3	Big	Run-of-mine
Washington	Pierce	Wilkeson-Wingate	No. 4 east dip	1⅝x0 in.
Washington	Thurston	Tono	Tono	1⅝x⅛ in.
Washington	Whatcom	Bellingham No. 1	Bellingham	4x2 in.
Wyoming	Sweetwater	D. O. Clark	No. 9	Run-of-mine

[a] All coal samples were raw except those from the King, McKay, and Bellingham mines.

tained for a particular coal. They are: 1—The clearance between the wearing blades and the mortar; 2—the speed at which the blades are rotated; 3—the weight of the coal charge in the mortar; 4—the duration of the test period; and 5—the particle size of the coal tested.

If the wearing blades used in the test procedure are rotated in a charge of coal held in a container substantially larger than the mortar normally employed, little wear of the blades takes place. The crowding or squeezing action between the outside edges of the blades and the wall of the mortar is responsible for the abrasion sustained by the blades. Therefore, the clearance between the blades and the wall of the mortar is critical in determining the magnitude of the abrasion loss. If the clearance is too great, the abrasion loss is decreased; but if the clearance is too small the abrasion loss is likewise decreased. The optimum clearance to produce maximum abrasion doubtless is influenced by the maximum size of particles in the coal charge.

Table II shows the results of abrasion tests made on four coal samples with clearances between the blades and the wall of the mortar ranging from 0.10 to 0.30 in. These tests were made on coal having a top size of 4-mesh, or 0.185 in. It will be noted that, with all samples, the abrasion loss increased steadily with increase in clearance. Maximum abrasion loss with most coals probably would occur with a clearance of about 0.30 in. Nearly all of the results in this report were obtained in tests made with a side clearance of 0.20 in., and, consequently, do not represent the maximum abrasion loss that could be achieved.

Table II. Influence of Side Clearance on Abrasion Loss

Coal	Side Clearance, In.	Abrasion Loss Mg.	Abrasion Loss Pct[a]
Olson-Cumberland	0.10	16	22
	0.15	62	83
	0.20	75	100
	0.25	82	109
	0.30	82	109
Roslyn No. 3	0.10	63	35
	0.15	151	85
	0.20	179	100
	0.25	199	111
	0.30	205	115
Montour No. 10	0.10	36	26
	0.15	96	69
	0.20	140	100
	0.25	141	100
	0.30	138	98
McKay	0.10	7	40
	0.15	14	74
	0.20	18	100
	0.25	22	122
	0.30	22	122

[a] Values expressed as percentage of abrasion loss obtained with clearance of 0.20 in.

The clearance between the bottom edges of the blades and the bottom of the mortar also influences the magnitude of the abrasion loss, as demonstrated in Table III, which shows results obtained in a series of tests on Olson-Cumberland coal where the bottom clearance was varied from 0.06 to 0.30 in.

Maximum abrasion loss was obtained with a bottom clearance of 0.20 in.

The side clearance required adjustment only when new wearing blades were substituted for badly worn ones. When used in testing ordinary raw coals, a set of wearing blades will last 25 to 30 tests before being worn down enough to increase the side clearance significantly. Adjustment of the bottom clearance was necessitated only to compensate for wear of the brass thrust bearing, and this adjustment was not required oftener than about every 50 tests. Substitution of a ball-bearing unit for the thrust bearing would eliminate any attention to bottom clearance.

With the equipment available, the rotation of the wearing blades could be varied through the range from 600 to 1500 rpm. The magnitude of the abrasion loss was almost directly proportional to the speed of rotation. For this reason, 1500 rpm was selected as the standard speed. No attempt to operate at higher speeds was made, despite the fact that speeds greater than 1500 rpm probably would give higher abrasion losses.

The mortar will hold a maximum charge of about 4 kg of a light coal, and this weight of charge was adopted as a standard. The data in Table IV show that, in the range from 1 to 4 kg, the abrasion loss is roughly proportional to the weight of coal.

Four kg is sufficient weight to constitute a representative sample of 4-mesh to 0 size coal. Use of a smaller quantity of coal of this size would not only

Table III. Influence of Bottom Clearance on Abrasion Loss

Bottom Clearance, In.	Abrasion Loss, Mg.
0.06	76
0.09	81
0.13	82
0.20	99
0.25	97
0.30	87

Table IV. Influence of Weight of Feed Charge on Abrasion Loss

Coal	Weight of Charge, Kg	Abrasion Loss Mg.	Abrasion Loss Pct[a]
Roslyn No. 3	1	43	23
Roslyn No. 3	2	75	41
Roslyn No. 3	3	146	78
Roslyn No. 3	4	186	100
Langley No. 9	1	58	25
Langley No. 9	2	108	46
Langley No. 9	3	196	84
Langley No. 9	4	234	100
Bellingham	1	38	32
Bellingham	2	57	47
Bellingham	3	93	78
Bellingham	4	120	100

[a] Values expressed as percentage of abrasion loss obtained with 4-kg charge.

Table V. Influence of Duration of Test on Abrasion Loss

Coal	Abrasion Loss, Mg. 6000 Rev.	Abrasion Loss, Mg. 12,000 Rev.	Increase, Pct[a]
Roslyn Big Bed	50	84	68
McKay	17	26	53
Wilkeson-Wingate	23	33	43
King	51	89	78
Roslyn No. 3	78	136	74

[a] Percentage increase over abrasion values obtained at 6000 revolutions.

decrease the abrasion loss obtained, but also militate against the representativeness of the sample.

Test durations of 6000 and 12,000 revolutions have been used. As shown by the abrasion losses presented in Table V, doubling the length of the test does not double the abrasion loss obtained. With the five coal samples used, doubling the duration of the test increased the abrasion loss by an average of 63 pct.

Most of the abrasion on the wearing blades is caused by the coarser particles of coal. Since the coal is progressively pulverized during the course of the test, fewer coarse particles are present to cause wear during the later stages of a test of longer duration. Thus, the grindability of a coal influences its performance in the abrasion test. A coal that is easily crushed is soon reduced to the finer sizes and exhibits less abrasiveness, while a tougher coal is not pulverized as rapidly and exhibits abrasiveness over a longer period.

This relationship is illustrated in Table V in the comparison between the samples of Wilkeson-Wingate and King coals. The Wilkeson-Wingate coal is easily pulverized, and doubling the duration of the abrasion test increased the abrasion loss only 43 pct. The King coal is more difficult to pulverize, and it showed a 78-pct increase in abrasion loss when the test period was doubled.

The influence of the grindability of a coal on its abrasiveness, as shown by the present test method, appears to be one of the disadvantages of the test procedure. However, this disadvantage could not be overcome readily in any other testing method except one in which the size composition of the abrading coal was kept constant by the continuous addition of coarse particles and removal of fine material. This is a cumbersome test condition, difficult to establish.

Table VI. Influence of Particle Size Tested on Abrasion Loss[a]

Screen Size, Mesh	Size Tested 4-Mesh to 0 Before	Size Tested 4-Mesh to 0 After	Size Tested 8-Mesh to 0 Before	Size Tested 8-Mesh to 0 After	Size Tested 14-Mesh to 0 Before	Size Tested 14-Mesh to 0 After	Size Tested 28-Mesh to 0 Before	Size Tested 28-Mesh to 0 After
4 to 8	32.3	6.0						
8 to 14	31.1	21.2	45.9	24.1				
14 to 28	15.6	14.5	23.1	20.0	42.6	32.5		
28 to 48	9.3	8.3	13.8	13.3	25.4	23.0	44.3	33.5
48 to 100	5.9	5.4	8.7	8.1	16.1	15.4	28.1	28.1
100 to 200	3.1	6.2	4.5	5.1	8.5	7.0	14.7	16.5
Through 200	2.7	38.4	4.0	29.4	7.4	22.1	12.9	21.9
Abrasion Loss, Mg.	185		152		75		33	
Reduction in Average Particle Size During Test, Pct	69		38		19		17	

[a] Abrasion tests made on Roslyn No. 3 coal.

The extent to which abrasion loss is influenced by the size composition of coal present in the charge is indicated in Table VI. A series of abrasion tests was made on Roslyn No. 3 coal with the standard side clearance of 0.20 in. The top size of material tested was 4, 8, 14, and 28-mesh. The abrasion losses obtained for these different sizes of coal range from 185 mg for the standard 4-mesh to 0 size to 33 mg for the material having a top size of 28-mesh. The particles coarser than about 14-mesh appear to be responsible for most of the abrasion.

The screen analyses of the coal before and after each test indicate the amount of grinding that took place during the test. The amount of grinding may be expressed as percentage reduction in average particle size, which varied from 69 with the 4-mesh coal to 17 with the 28-mesh coal. Thus, there is an approximate correlation between the amount of abrasion obtained and the amount of grinding that takes place.

The rate at which coarse particles are ground to finer size and the influence of this change in size composition on abrasion loss are demonstrated by the results presented in Table VII. This table summarizes the results obtained in a test made on Roslyn No. 3 coal in which all test conditions were standard, except that the apparatus was stopped at the end of each 1000 rev, and the size composition and the abrasion loss determined for each 1000 rev. The percentage of 4 to 8-mesh coal was reduced sharply during the first 1000 rev and then decreased at a much slower rate throughout the remainder of the test. A similar trend is evident with the 8 to

Table VII. Rate of Abrasion and Grinding During Test Period[a]

No. of Revolutions	Abrasion Loss, Mg.	Screen Analysis, Pct 4 to 8 Mesh	8 to 14 Mesh	14 to 28 Mesh	28 to 48 Mesh	48 to 100 Mesh	100 to 200 Mesh	Under 200 Mesh
0		30.2	33.3	16.0	9.3	5.4	3.6	2.2
1,000	15	20.9	28.5	15.8	9.8	7.8	5.8	11.4
2,000	12	18.0	27.5	14.8	10.6	7.4	6.1	15.6
3,000	13	13.7	27.7	16.6	9.9	7.5	5.8	18.8
4,000	12	12.3	27.5	16.1	9.8	6.3	6.5	21.5
5,000	13	13.8	27.3	15.9	9.2	5.9	5.3	22.6
6,000	13	13.6	26.6	15.3	8.3	5.5	5.1	25.6
7,000	12	10.5	26.3	15.3	7.9	5.6	5.2	29.2
8,000	10	10.9	25.6	14.9	8.3	4.9	5.2	30.2
9,000	9	10.2	25.1	15.3	7.9	5.2	5.2	31.1
10,000	9	10.2	24.9	14.6	7.8	5.0	5.4	32.1
11,000	8	9.1	26.0	15.1	7.4	4.6	4.8	33.0
12,000	10	8.8	24.9	14.5	7.6	4.7	4.7	34.8

[a] Test made on Roslyn No. 3 coal.

Table VIII. Screen Analyses of Soft and Hard Coals Before and After Abrasion Testing

	Wilkeson-Wingate Float 1.40		Pennsylvania Anthracite	
Size, Mesh	Before Test, Weight, Pct	After Test, Weight, Pct	Before Test, Weight, Pct	After Test, Weight, Pct
4 to 8	42.2	1.0	57.7	24.9
8 to 14	35.9	11.2	22.8	23.4
14 to 28	13.6	14.8	9.7	12.1
28 to 48	4.6	10.1	5.0	7.1
48 to 100	2.2	6.7	2.8	4.5
100 to 200	1.0	9.4	1.4	5.0
Under 200	0.5	46.8	0.6	23.0

Table IX. Relation Between Abrasion Loss and Moisture Content

Moisture Added, Pct	Total Moisture, Pct	Abrasion Loss, Mg.
0	3.1	163
.5	3.6	186
1.0	4.1	190
3.0	6.1	216

14-mesh material, although this size fraction was not depleted as much as the 4 to 8-mesh fraction. The proportions of the size groups between 14 and 200-mesh did not change significantly during the entire course of the test.

During the first 1000 rev the wearing blades lost 15 mg, and wear continued only at a moderately decreased rate during the next 6000 rev. The rate of abrasion fell somewhat after 7000 rev, but not sharply.

Roslyn No. 3 coal has a Ball-Mill grindability index of 35 pct, and thus is moderately difficult to grind. Naturally, with coals of high grindability, the coarser sizes are eliminated more rapidly than the rate indicated for Roslyn No. 3. Conversely, the coarse sizes disappear more slowly with coals of low grindability. The influence of grindability on the amount of coarse material remaining at the end of an abrasion test is shown in Table VIII, which gives the screen analyses of the float 1.40 sp gr fraction of Wilkeson-Wingate coal and a Pennsylvania anthracite before and after testing.

Table X. Results of Abrasion Tests Showing Precision of Test Procedure

	Abrasion Loss, Mg.	
Coal	New Blades	Worn Blades
King	162.6	174.7
King	158.4	176.6
King	159.2	177.3
King	161.3	182.2
Average	*160.4*	*177.7*
Wilkeson-Wingate float 1.60 sp gr	16.6	16.5
Wilkeson-Wingate float 1.60 sp gr	13.8	17.5
Wilkeson-Wingate float 1.60 sp gr	14.7	17.6
Wilkeson-Wingate float 1.60 sp gr	15.6	15.8
Average	*15.2*	*16.9*
Olson-Cumberland	87.6	
Olson-Cumberland	89.2	
Olson-Cumberland	87.7	
Olson-Cumberland	90.0	
Average	*89.2*	

With the Wilkeson-Wingate coal, which is easily pulverized, 12 pct of material coarser than 14-mesh remained at the completion of the test, while with the stronger Pennsylvania anthracite 48 pct of this size remained. The principal consideration in fixing the duration of the abrasion test at 12,000 rev was to insure that, with even the softest coal, some coarse material would be present in the charge at the end of the test.

Some of the discrepancies in the results of earlier abrasion tests directed attention toward the possible effect of the moisture content of the coal tested. Accordingly, a series of samples of King coal of different moisture contents was prepared by adding predetermined amounts of water to air-dried coal. After the moisture was added, the samples were placed in sealed containers, mixed well, and then allowed to stand overnight to insure a uniform distribution of moisture. Table IX shows the influence of these moisture additions on abrasion loss.

The air-dried coal, containing 3.1 pct moisture, gave an abrasion loss of 163 mg; addition of only 0.5 pct moisture increased this loss to 186 mg; and addition of 3 pct moisture increased the loss to 216 mg. Thus, the presence of even a small amount of excess moisture increases abrasiveness significantly.

With a change in moisture content of only 0.5 pct having such a marked effect on abrasion loss, it might be expected that the moisture change in air-dried coal due to changes in atmospheric humidity might affect abrasion results. But, samples of several coals have been tested repeatedly throughout several months of laboratory storage with consistent abrasion results.

The role played by moisture in the abrasiveness test is not clear. One explanation would be that the presence of even a small amount of water in the pores of the coal toughens it considerably. Under the conditions existing in the test procedure, corrosion resulting from the presence of the added water seems unlikely.

Precision of Test Method

The results of abrasion tests made in quadruplicate on three samples of coal are shown in Table X to demonstrate the degree of precision of the test procedure. The float 1.60-sp gr fraction of the Wilkeson-Wingate coal was selected to provide a material of low abrasiveness, the Olson-Cumberland coal is moderately abrasive, and the King coal is one of high abrasiveness. In the tests made with new wearing blades, the maximum difference between the four tests on Wilkeson-Wingate coal was 2.8 mg. The corresponding deviation for the samples of Olson-Cumberland and King coal was 2.4 and 4.2 mg, respectively. The 4.2 mg maximum deviation for the King coal represents 2.6 pct of the average abrasion value for this coal. This percentage deviation is probably no greater, for example, than that inherent in the standard friability test for coal. The 2.8 mg maximum deviation for the less abrasive Wilkeson-Wingate coal, however, amounts to 18.4 pct. Thus, with coals giving low abrasion losses, the precision of the method is lower than desirable.

The data in Table X also show a comparison between the abrasion results obtained with new and worn blades. The worn blades had been used long enough to sustain a total loss in weight of over 5 g. Ordinarily, blades are discarded after a total loss of 3 or 4 g, so the comparison shown is extreme. With both the King and Wilkeson-Wingate coals,

Table XI. Results of Abrasion Tests Made with Various Metals

Coal	Armco Iron	Mild Steel 1020	Type 301 Stainless Steel		Type 410 Stainless Steel
			Annealed	Full Hard	Annealed
Olson-Cumberland	89	79	71	78	78
Roslyn No. 3	182	162	137	146	141
King	162	149	131	144	140
Rockwell hardness B scale	42	62	88	118[a]	82

[a] A scale 75, roughly equivalent to B scale 118.

the badly worn blades gave abrasion results averaging about 11 pct higher than those obtained with new blades. Thus, if blades are discarded after being used for making enough tests to cause losses totaling not over 2 or 3 g, the influence of blade wear is probably within the accuracy of the testing procedure.

Although the accuracy of this test procedure, particularly with coals of low abrasiveness, is not as high as might be desired, it probably is good enough for the industrial application for which the method is intended. In other words, the industrial counterpart of the abrasion test, the wear of pulverizer grinding units, cannot be determined with great accuracy, and therefore a precise testing procedure probably is not required.

Comparison of Various Metals

Armco iron was originally adopted as the standard metal for the wearing blades for two reasons. First, it was thought that, being relatively soft, Armco iron would give higher abrasion losses. Second, Armco iron reputedly does not work harden and thus should offer constant resistance to abrasion. Comparison of the abrasion resistance of various metals is a complete metallurgical problem in itself and is outside the scope of the present investigation. However, SAE 1020 mild steel and several stainless steels in both the annealed and full-hard condition were tested to show how these materials compare with the Armco iron used as a standard. Table XI shows the results of tests made on three kinds of coal with wearing blades constructed of five different metals.

Surprisingly, SAE 1020 mild steel gave abrasion losses only moderately lower than those obtained with the standard Armco iron. The stainless steels likewise did not show the superior resistance to abrasion that had been expected. An interesting feature of these data is that the type 301 stainless steel was less resistant to abrasion in full-hard form than when annealed.

No significance is placed on the data shown in Table XI except that they appear to indicate that the use of Armco iron as a standard material for wearing blades does not result in abrasion values that are out of line with those obtained with the more common ferrous alloys.

Since work hardening of the wearing blades, if it occurred, would change their resistance to abrasion progressively, Rockwell hardness tests were made on several sets of blades before use, after the blades were about half worn out, and after the blades had been worn so badly that they were ready to be discarded. No increase in R hardness could be detected. Similar tests were made on the stainless steel blades. Although these blades were used for only a few tests and could not be tested after the extensive use given Armco iron blades, no work hardening was evident with these blades either.

Table XII. Results of Abrasion Tests on Various Kinds of Coal

Coal	Abrasion Loss, Mg.
Cush Creek	12
McKay	18
Wilkeson-Wingate	42
Tono	45
Olson-Cumberland	92
Bellingham	120
D. O. Clark	130
King	165
Montour No. 10	171
Roslyn No. 3	185
Castle Gate	212
Langley No. 9	242
Pennsylvania Anthracite	686
Sandstone	1210
Coke	2506

Abrasion Tests

Table XII summarizes the results of abrasion tests made on all the coal tested under standardized conditions. These standardized conditions were: Rotation of 1500 rpm; 12,000-rev duration of test; side and bottom clearance of 0.20 in.; Armco iron wearing blades; and 4-mesh to 0 coal.

Excluding the Pennsylvania anthracite, which gave an abrasion loss of 686 mg, these coals ranged in abrasiveness from 12 to 242 mg. In general, there is no direct correlation between the abrasiveness of coal and either its rank or its ash content in the raw state. The two strongly-caking samples of coal, Cush Creek and Wilkeson-Wingate, both gave abrasion losses of less than 50 mg. Whether or not low abrasiveness characterizes all strongly caking coal, however, scarcely can be deduced from tests of only two such coals. The Pennsylvania anthracite was

Table XIII. Results of Abrasion Tests Made on Specific-Gravity Fractions of Various Kinds of Coal

Coal	Specific Gravity	Weight, Pct	Ash,[a] Pct	Abrasion Loss, Mg.	Cumulative		
					Weight, Pct	Ash,[a] Pct	Abrasion Loss, Mg.
Wilkeson-Wingate	Under 1.30	11.5	4.9	4	11.5	4.9	4
	1.30 to 1.40	37.4	10.9	5	48.9	9.5	6
	1.40 to 1.50	12.6	19.1	30	61.5	11.5	10
	1.50 to 1.60	5.3	25.9	54	66.8	12.6	13
	Over 1.60	33.2	61.6	289	100.0	28.9	105
	Total raw coal						43
Tono	Under 1.30	41.0	5.9	17	41.0	5.9	17
	1.30 to 1.40	45.9	11.6	38	86.9	8.9	28
	1.40 to 1.50	8.7	27.8	54	95.6	10.6	39
	Over 1.50	4.4	56.6		100.0	12.7	
	Total raw coal						45
Langley No. 9	Under 1.60	90.7	9.3	45	90.7	9.3	45
	Over 1.60	9.3	58.3	1515	100.0	13.8	181
	Total raw coal						234
Pennsylvania Anthracite	Under 1.80	81.1	7.6	63	81.1	7.6	63
	Over 1.80	18.9	71.8	2847	100.0	19.8	589
	Total raw coal						686
Cush Creek	Under 1.60	92.9	5.6	6	92.9	5.6	6
	Over 1.60	7.1	62.1	351	100.0	9.6	12
	Total raw coal						12
Montour No. 10	Under 1.60	79.3	9.1	43	79.3	9.1	43
	Over 1.60	20.7	75.9	618	100.0	22.9	162
	Total raw coal						172
Castle Gate	Under 1.60	95.2	6.7	147	95.2	6.7	147
	Over 1.60	4.8	63.7	1517	100.0	9.4	213
	Total raw coal						212

[a] Moisture-free basis.

over three times as abrasive as most bituminous coal; but, as will be shown later, this abrasiveness was caused more by the character of the impurities associated with it than by the coal substance itself.

For the sake of comparison, the results of abrasion tests on crushed sandstone and coke are included with the coal in Table XII. The sandstone gave an abrasion loss of 1210 mg, and the abrasiveness of the coke was 2506 mg. Thus, both of these materials, which are highly abrasive, are shown by this test procedure to be many times more abrasive than coal.

Tests of Specific-Gravity Fractions

To determine to what extent the abrasiveness of a raw coal is caused by the character of the coal substance itself and to what extent it is caused by the impurities associated with the coal, five coal samples were separated into clean coal and impurity by float-and-sink tests at 1.60 sp gr. Abrasion tests were then made separately on the fractions. Two other samples of coal were separated at additional specific gravities to show the character of the material of intermediate density. Table XIII presents the results.

In every instance, the impurity heavier than 1.60 or 1.80 sp gr was much more abrasive than the clean coal. For example, clean Pennsylvania anthracite lighter than 1.80 sp gr gave an abrasion loss of 63 mg, while the heavy impurity gave 2847 mg. This impurity, which appeared to be slate, was the most abrasive material tested, exceeding in abrasiveness even the figure of 2506 mg for coke. The float 1.80 sp gr fraction of the Pennsylvania anthracite was less abrasive than many raw coals tested, and less abrasive than even the float fraction of the Castle Gate coal.

It will be observed that, with several of the coal samples shown in Table XIII, the cumulative abrasion loss for the specific gravity fractions tested individually differs appreciably from the abrasion loss obtained when these materials were tested combined together as raw coal. The explanation for this discrepancy is not the same for all coals. With the Langley No. 9 coal, the cumulative abrasion loss of 181 mg is thought to be less than the value of 234 mg obtained on the raw coal because of the loss of considerable clay during the float-and-sink manipulation. With the Wilkeson-Wingate coal, however, in which the cumulative abrasion loss is 105 mg in comparison with a loss of 43 mg for the total raw coal, the explanation appeared to be that, in tests of the raw coal, the wearing blades were shielded from action of the abrasive sink material by the presence of the much less abrasive clean coal.

The results presented in Table XIII for seven samples of coal demonstrate that abrasive wear can be attributed more to the impurities associated with coal than to the character of the coal substance itself. Cleaning should almost invariably reduce the abrasiveness of a coal through the removal of abrasive impurities, and clean, well-prepared coal will cover a much narrower range of abrasiveness than that exhibited by unwashed coal.

Summary and Conclusions

A laboratory procedure for estimating the abrasiveness of coal was developed, in which a set of wearing blades is revolved for a fixed number of revolutions in a charge of coal. The loss in weight sustained by the blades is taken as a measure of the abrasiveness of the coal.

As with any empirical test method, the results are influenced by test conditions, and such variables as speed and duration of rotation, weight and size of coal employed, and clearance between the wearing blades and the walls of the mortar containing the coal sample were investigated. With such variables properly controlled, the method gives results that can be reproduced within about 3 pct with abrasive coals, with variations of up to 18 pct occur in testing coals of very low abrasiveness. This degree of precision probably is sufficient for most purposes.

Fourteen coals have been tested, and seven of these were separated into clean coal and impurity by float-and-sink procedure to permit testing the coal and its associated impurities separately for abrasiveness. In every case the impurities proved far more abrasive than the coal substance itself. Clean coal ranged in abrasiveness from about 10 to 150 mg, while the heavy impurities separated from these samples ranged from about 300 to 2850 mg. Thus the amount and character of the impurities associated with a coal have a marked influence on its abrasiveness.

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New Mining Methods Tested by Menominee Range Iron Ore Producers

by Philip D. Pearson and Warren W. Jamar

IN recent years, there have been many changes in mining operations in the Lake Superior district. To follow these trends on the Menominee Range of Michigan, information has been assembled from all of the iron mining operations in the area. There are 15 operating underground mines in the Iron River-Crystal Falls area of the Menominee Range. Within the past two years, two idle properties were reopened and are now producing, and a third and fourth are being reopened. Also, there are two siliceous openpits operated by independent companies outside this immediate area. Six companies operate the underground mines, employing some 1850 employees. Table I shows pertinent facts about these properties.

During 1949, the largest mine in Iron County shipped 571,287 tons, and one of the newer mines shipped 39,378 tons, with a range total to 3,535,373 tons. Since the Menominee Range was opened in the 1870's, the mines in Iron County have shipped 85,890,922 tons.

From an operator's viewpoint rather than a geologist's, the ore is classified as semi-hard, composed of hematite and limonite. It is not as soft as the ores of the Marquette Range nor is it as hard as the hard ores of the Marquette and Vermillion Ranges. The ore bodies have slate hanging walls and slate footwalls. In most cases the hanging walls and footwalls are soft and high in sulphur. The sulphur comes from pyrite, and these slates will ignite when piled more than 6 to 8 ft high.

Ore is mined on this range by: 1—Sub-level stoping; 2—Shrinkage stoping; 3—Sub-level caving; 4—Block caving; 5—Top slicing. The predominance of these methods is in the order named.

Underground drilling is important in the mining cycle. Some changes made and trends toward future changes fall into four categories: Drill bits, drill steel, drill machines, and compressed air pressures. Several types of bits have been tried and are in use. They include the detachable tungsten carbide, insert bit; the intraset steel bit, which is a conventional steel rod with tungsten carbide insert; the one-use bit; and the multiple-use bit.

For many years, detachable multiple-use bits have been standard. In tests conducted recently to improve drilling efficiency, this bit was used as the basis for comparison. Under existing conditions, a multiple-use bit can be resharpened about three times before it is discarded.

A thorough test of 2-in. tungsten carbide threaded bits was conducted under various ground conditions. 1—The drilled footage ranged from 48 to 600 ft per bit; averaging 357 ft per bit. Under these same conditions, a multiple-use bit ranged from 8 to 80 ft per bit. 2—The bit cost was greater for the insert bit in each case. 3—The average drilling speed for the insert bit was 12 in. in 62 sec and for the multiple-use bit was 12 in. in 64 sec.

In a second test, 2¼-in. tungsten carbide bits with the large 1 3/16-in. thread were used in moderately soft ground on a 152-lb drifting drill on a long feed jumbo. 1—The drilled footage ranged from 450 to 5000 ft per bit, averaging 1810 ft per bit. Under these same conditions, a multiple-use bit averaged 64 ft per bit. 2—The bit cost was reduced by the use of the insert bit. 3—No increase in drilling speed was recorded. 4—Minimum footage obtained was caused by thread failure. To improve the thread life, thread size on the rod was increased and the bit was attached to the rod with a pipe wrench. After this, bits failed in equal proportions because of cracked skirts, broken inserts, and gage loss.

Tungsten carbide bits are now used in the operation where this second test was conducted because labor costs were lowered as a result of reducing the number of bits being changed by the miners. At the same operation and under the same conditions as the second test, 1¾-in. insert bits with standard 1-in. threads were used. These bits did not drill much more than 300 ft before thread failure and were then welded to the rods and used until total failure. Sometimes this footage was considerable, sometimes it was not. Chisel-type 2¼-in. insert bits were

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found unsatisfactory for two reasons. Thread life was bad, and the bit tended to penetrate too far into the material being drilled and therefore could not be turned by the machine. An overall loss in drilling speed resulted.

Another operator who at present uses insert bits in rock only reported the following: Minimum footage, 236 ft per bit; maximum footage, 905 ft per bit; average, 402 ft per bit. The usual problem of threads was encountered, but thread failure was lessened by close supervision of both the rod and the bit. *Go* and *No-Go* gages for both rods and bits are furnished to miners and shift bosses, and these men check the rods and bits frequently.

At another mine, drilling in a shaft sinking operation was done with insert bits. Cross and chisel type 1¾ and 1⅝-in. insert bits were used and averaged 190 ft per bit. Thread life was the determining factor, and the chisel-type bits held up longer than cross bits. A multiple-use bit, under the same conditions, drilled about 8 ft per bit. The insert bit cost less than the multiple-use bit cost, and considerable time was saved because it was not necessary to change bits while drilling a cut. All bits were put on the rods in the shop and the threads were inspected. They were rethreaded, if necessary, before strippage occurred. Several operators experimented with attaching the threaded insert bits to the rods with silver solder.

A description of this process, as given by one of these operators, is as follows: The thread of the rod is cleaned in a shot blast and with a wire brush. The bit thread is cleaned as well as possible with a brush. Two holes are drilled in the bit body opposite each other and near the bottom of the thread. The thread end of the rod is heated with a torch to a dull red, and the bit is screwed on with a 24-in. pipe wrench. The skirt of the bit is heated uniformly until the solder flows (about 1200°F). Care must be taken not to overheat the wings of the bit since the inserts are brazed in and will fall out if the weld is overheated. Easy-flow solder in the form of wire is applied in one of the holes until it flows out of the other hole, indicating that any spaces between the threads are filled. During the application of the solder, heat is applied with the

Table 1. Data About Operating Mines on the Menominee Range

No.	Depth of Shaft, Ft	1949 Shipments	Number Employed	Mining Methods	Tramming Method	Crushing Method	Stockpiling Method	Skip Loading Method	Distance Between New Levels, Ft	Pumping Capacity, Gpm	Skip Size, Tons
1	1100	527,666	129	Sub-level stope	Tram car and locomotive	Gyratory-underground	Dumptor	Storage pocket and measuring pocket	200	100	4
2	1100	393,029	184	Sub-level stope	Tram car and locomotive	Gyratory-underground	Dumptor	Storage pocket and measuring pocket	200	1400	4
3	2300	300,603	139	Sub-level stope	Tram car and locomotive	Gyratory-headframe	Locomotive and tram car on stockpile	Car — skip	200	500	4
4	2300	268,699	135	Sub-level stope	Tram car and locomotive	Gyratory-headframe	Larry car on stockpile	Car — skip	200	300	4
5	600	39,378	60	Sub-level stope	Tram car and locomotive	Gyratory-headframe	Endless rope and trestle	Car — skip	200	350	4
6	1100	571,287	323	Sub-level stope	Tram car and locomotive	Gyratory-headframe	Endless rope, locomotive and cars and trestle	Trench and measuring pocket	200	260	3
7	1050	273,339	166	Sub-level	Tram car and locomotive	Jaw-underground	Endless rope and dozer dumptor (rock)	Trench and measuring pocket	200	120	3
8	950	191,529	120	Sub-level stope	Tram car and locomotive	Jaw-underground	Endless rope and bulldozer	Rotary dump, storage and measuring pockets	200	130	3
9	1300	New property				Headframe					
10	1300	420,718	97	Sub-level stope	Belt conveyor	Gyratory-headframe	Dumptor	Storage pocket and measuring pocket	250	100	6
11	1650	New property	70	Shrinkage stope	Tram car and locomotive, belt conveyor	Gyratory-headframe	Dumptor	Trench and measuring pocket		100	5
12	1800	346,286	182	Sub-level stope	Tram car and locomotive, belt conveyor	Gyratory-headframe	Larry car and trestle and bulldozer	Rotary dump, car to skip		400	4
13	550	114,386	75	Sub-level stope	Tram car and locomotive		Dumptor	Trench and measuring pocket	200	900	3
14	1250	New property	10	Exploration					150	10	
15	1000	88,453	116	Sub-level and shrinkage stope	Tram car and locomotive	Gyratory-headframe	Locomotive and cars, trestle and bulldozer	Trench and measuring pocket	350	750	5
16		36,607		Open-pit							
17		14,101		Open-pit							
		3,586,081									

Table II. Results of Tests on One-Use Bits

	152-lb Post-Mounted Drifting Drill		127-lb Post-Mounted Drifting Drill		55-lb Hand-Held Drill	
	In. per Bit Use	In. per Min	In. per Bit Use	In. per Min	In. per Bit Use	In. per Min
2⅛-in. Multiple-use bit	30	8.5	39	8.5	60	9.0
2⅛-in. One-use bit	60	8.25	72	8.0	132	9.0

torch to keep the temperature even around the periphery of the skirt. The bit is allowed to cool in air.

The intraset steel bits are being tested. Early results of the tests of one operator were encouraging, averaging approximately 840 ft per rod total life in ground where a multiple-use bit drilled 20 ft per bit. These rods were 1¼-in. rod alloy steel and drilled 2¼ and 2⅛-in. holes. It appears that intraset steel bits may be used profitably, but until tests are completed and more information is available, specific conclusions cannot be set forth.

Three types of one-use bits have been tested at one mine on the Menominee Range. Testing of these bits was complete and results were gratifying. In these tests, three machines were set up in an old stope, and holes were drilled about 6 in. apart, so that for all practical purposes the ground was similar for each bit. Records of air pressure, throttled time, depth of hole, and footage per bit were kept. All told, 400 to 500 bits were used. The results of these tests are shown in Table II. After a production test by using the one-use bit in the actual operation, this operator recently changed over to the one-use bit.

It appears that the tungsten carbide bit on the Menominee Range has no real advantage over the multiple-use bit or the one-use bit. In hard ground, the tungsten carbide bit has an advantage in time and cost; but in the usual Menominee Range ore there is no advantage. Recent tests show that the one-use bit may have an advantage over both the standard multiple-use bit and the tungsten carbide bit. The intraset bit is still being tested and has been adopted for limited uses.

While the various types of drill bits have been well tested, in contrast it appears that the various types of drill rods have not had complete tests. Since bit cost and rod cost per ton mined are related, further tests and improvement on the drill rods can be made.

About 2 or 3 years ago, operators tested alloy steel drill rods. Some changed over completely to the alloy steel rod. At one mine, records of the results with alloy steel as compared to carbon steel were kept. These records showed that, when made up in the mine shops and tested on a block tester and underground, alloy steel rods were about 60 pct as good as carbon steel rods. Alloy steel rods made up at outside sources proved only 30 pct as good as carbon steel rods.

Carbon drill steel, which has been and still is being used at the majority of mines on the Menominee Range, has undergone little change. However, improvements have been in the form of better and closer control. In this regard, one operator obtained the following results: 1-in. hexagon steel, used on a 127-lb drifting drill with 80-lb air pressure (machine running), gave a minimum drilling time of 35 min to a maximum of 250 min, averaging 125 min per rod. Breakage of rods at the minimum time took place at the collar; breakage at the maximum time took place about 6 to 8 in. in front of the collar. Drilling speed averaged about 6 to 8 ipm. Prior to this test, rods had been tested that gave a life of from 10 to about 100 min per rod.

The life of a piece of drill steel after it has had a second shank or a second thread is materially reduced. From this, the question arises as to whether or not it is worthwhile to reshank and rethread.

It appears that little improvement has been made in drill steel, with the exception of better control in heat-treating processes. The introduction of alloy steel has not been successful on this range, and the trend is away from rather than toward it. Since bit cost and rod cost per ton are approximately the same, it would appear advisable to continue to experiment in the hope of finding something better than the carbon steel drill rod.

Important also to the drilling cycle are drilling machines. Almost all manufacturers have improved standard machines or introduced new designs during the past few years. One of the most effective changes was the increase in the length of feed on machines mounted on jumbos for main drifting. These long feeds reduce the time required in the overall drilling cycle. Table III shows the results of a time-study taken for machines with a 2-ft change on an old style, manually-operated jumbo, compared with machines with an 8-ft change mounted on a new style automatically-operated jumbo. The introduction of insert bits helped make possible this change in machines.

A recently opened property adopted as standard in the shrinkage stopes a reverse, air-fed, post-mounted machine. This type is attached to an arm and moved by air pressure and has found application in long hole drilling. It has replaced, to a small extent for specific operations, the hand-held drill on jack legs and the post-mounted drifting drill. Its advantages are: Rod changing is faster, less physical

Table III. Time Study of Drilling Cycle with Two Types of Jumbos

Operation	Old Style Time, Min 2-ft Change — 6-ft Holes	New Style Time, Min 8-ft Change — 8-ft Holes
Setup	20	15
Aligning between holes	65	40
Throttle time	75	80
Tear down	5	5
Total	165	140

Table IV. Drilling Speed vs. Air Pressure

Pressure, Psi	Rate of Penetration in Hard Ground, Ipm	
	3½-in. Power Feed Drifter	2⅜-in. Jack Hammer
60	—	4.3
70	4.8	—
75	5.1	5.5
85	8.0	7.7
90	9.2	9.0
100	11.5	10.3
105	—	10.6 (?)

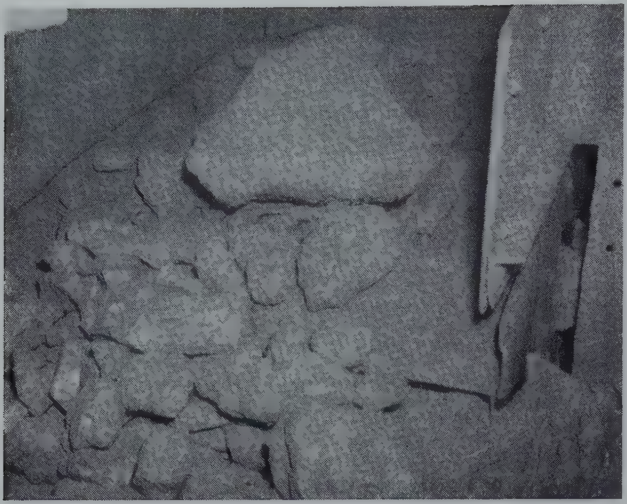

Fig. 1—Large pieces are hauled to the underground crusher.

work is required, and long holes can be drilled accurately.

The various sized hand-held drills on jack legs are used to advantage in several operations on this range, and drifting drills on posts are used in a few operations.

Drilling speed can be increased by increasing air pressure at the drill machine. This is being done by one large operator by installing the proper sizes of pipes and increasing the air pressure at the compressor. Table IV indicates the increase in drilling speed as the pressure rises. In conjunction with this increased air pressure it is claimed that a reduction in bit and rod cost per foot of drilled holes can be realized. However, no proof of this is available.

Transportation

There are two schools of thought on the Menominee Range as to which system of underground haulage best suits conditions. One is a system of handling large pieces with large locomotives and large tram cars; and the other to reduce the size of material and move it by belt conveyor. Neither system is advocated because there is no conflict. There is a proper place for each.

Where material is harder than normal, operators veer toward the large cars and locomotives. The method of mining at one operation produces large pieces of material. Little effort is made to reduce chunks to a size below 1x2x2 ft, the theory being that no time should be spent in secondary blasting. The material is conveyed to the shaft in 8-ton cars drawn by an 8-ton locomotive and is dumped into a crusher installed at the foot of the shaft. After crushing, the material is stored in a storage pocket, flowing by gravity into measuring pockets and thence into the skip. The loading of the skip is facilitated in that sized material is handled. The skip operator remains on the level to operate loading of skips. Fig. 1 shows large size material entering underground crusher.

Table V. A Haulage System in Operation on Menominee Range

	Conveyors		
	No. 1	No. 2	No. 3
Width of belt, in.	36	36	36
Center to center end pulleys, ft	818	648	635
Capacity, tons per hr	250	250	250
Speed, ft per min	200	200	200
Ac motors, hp	20	20	20
Lift, tail to head pulley, ft	2.1	4.8	5.5

The advantages of this system are: 1—Secondary blasting is minimized. 2—The grizzly, a dangerous operation, is not necessary. 3—Skip loading is easy and the possibility of large chunks being caught in the gates is eliminated, thereby improving the hoisting cycle. 4—The capital required to install a locomotive-tram car haulage system is approximately one third less than the amount required for a belt conveyor installation. 5—It is not necessary to drive the haulage-way in a straight line, as with a system of belt conveyors. 6—It is not necessary to have large concentrations of ore in one area as with a belt conveyor. 7—It is not necessary that 25 ft of ore in vertical distance be tied up because of the grizzly sub used in a belt system.

The size of the material being mined governs the choice of system. The length of haul is also a determining factor in making the choice between tram cars and belt conveyors. In long hauls the capital expenditure is proportionately much greater for belt equipment.

There is a trend on this range toward belt conveyor haulage systems. Fig. 2 shows a typical underground belt installation. In this system, 36-in. wide belts are used in place of the locomotive-tram car system. Table V describes the main haulage system of three belts used at one property. In addition, Table VI shows the amount of material moved to date by these belt conveyors.

In the belt conveyor system, material drops from the stope onto a rail grizzly. Chunks are reduced to a size below 14x14x9 in. Material then drops to a chute 25 ft below, where it is loaded onto a shaker conveyor and conveyed to a 36-in. wide main haulage belt. Fig. 3 shows a shaker conveyor feeding a belt conveyor. The belt conveys the material to the shaft where it drops into a storage pocket, and thence into measuring pockets to be loaded into the skips and hoisted to the surface. The crusher in this system is in the headframe on surface.

The advantages of the belt haulage system are: 1—Horsepower requirements per ton are much less. Table VII verifies this statement. 2—Fewer men

Table VI. Tons of Material Conveyed by Shaker Conveyor and Belt Conveyor at One Property

	Shaker Conveyors, Tons	Belt Conveyors, Tons
1943	59,000	
1944	107,000	
1945	92,000	
1946	90,000	2,500
1947	167,000	120,862
1948	462,524	462,524
1949[a]	387,254	387,254
1950 (Through October)	212,690	212,690
	1,577,468	1,185,830

[a] Total production lower because of six weeks' strike.

Table VII. Power Requirements per Ton of Ore Conveyed

	6-Ton Locomotive and Mine Cars 5-ton Capacity	36-In. Belt Conveyor
Tons per hr	100 max.	250
Tons per kwh	2.4	8.3
Kwh per ton	0.42	0.12
Power cost per 1000 tons ($0.013 per kwh)	5.46	1.56

are required to operate belt conveyors. 3—Belt conveyors are less hazardous than the locomotive-tram car system in tramming. 4—Transportation costs are lower.

Under conditions where the ore is hard, where the ore has a tendency to break in large chunks, where the orebodies are small and scattered, and the haulage road is extremely long, there is no question that the method of large cars, handling large material, is preferable. Under conditions where a large concentration of ore occurs in a comparatively small area, and where the character of the ground is such that there is not a predominance of large chunks and a necessity for secondary blasting, the belt conveyor system is preferable.

Normally, in an underground operation, after hauling material from the working place to the shaft, the next operation is skip loading. On the Menominee Range, there are four methods being employed to load material into skips. These methods are: Dump direct, scraping trench, storage bin-measuring pocket, and storage bin-measuring pocket and crusher.

In the dump direct method, ore is dumped from the car into a skip by a slide. In some cases a separate slide for each skip is used, and in other cases a slide with a hinged gate is used to deflect the material to each skip.

In the scraping trench method, ore is dumped into a trench from tram cars or from belt conveyors. The material is then scraped by an electric hoist into measuring pockets. Each pocket holds one skip, and the material is loaded directly from these pockets into the skips.

In the storage bin-measuring pocket method, the ore is dumped into a storage bin with a capacity of about 200 tons. From this storage bin the ore flows by gravity into skip measuring pockets and then into skips.

In the storage bin-measuring pocket and crusher method, ore is dumped into a scraping trench and then scraped into a crusher, dumped directly into a crusher, or dumped onto a feeder which feeds a crusher. From the crusher, ore is fed by gravity into a storage bin and flows from this bin into measuring pockets and then into the skip.

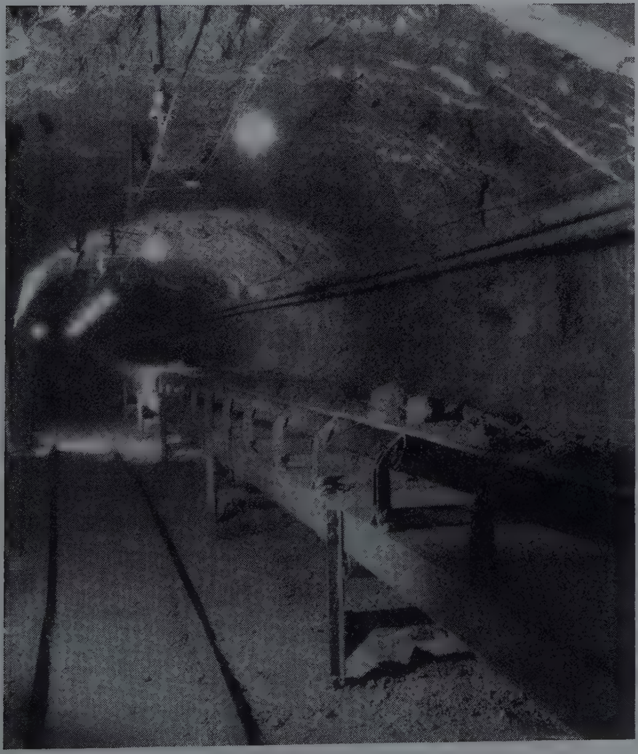

Fig. 2—Underground conveyor belt installation typical of those used on the Menominee Range.

Fig. 3—Shaker conveyor feeding a belt conveyor.

On the Menominee Range all these methods are used. Each has its advantages and disadvantages. If the ore is sticky and wet, the dump direct method or the scraping trench seems most satisfactory. If the ore is not sticky and is blocky, the storage bin-measuring pocket and crusher seems most satisfactory. If the ore is neither blocky nor sticky, the storage bin-measuring pocket seems to have advantages. The only trend is a greater use of the scraping trench, and this comes about where there is water in the ore.

There are several methods of surface stockpiling ore in use on the Menominee Range. These are: 1—Locomotive pulling a tram car supported by a stockpile trestle. 2—Locomotive pulling a tram car on rails supported by the stockpile material. 3—Larry-car supported by a trestle. 4—Endless rope system and wooden trestle. 5—Dumptor. A bulldozer is used to spread the material in several of these operations.

There appears to be a trend in the method of stockpiling ore on the Menominee Range. This trend is evidenced by the fact that since one operator in 1942 began stockpiling ore with a Dumptor, five others have adopted its use, and two others plan to adopt it.

With the Dumptor, ore is hoisted from underground in two 6-ton skips that dump directly into the crusher in the headframe. Crushed ore falls into a pocket large enough to hold one skip. This pocket is emptied through an air cylinder-operated gate into the body of the Dumptor. The Dumptor operator opens the gate when his truck is spotted and closes it when the 6-ton load has been discharged. Each skip makes a separate load for the Dumptor. This operation also could be carried on from a storage pocket. In that case it would not be necessary to tie in the Dumptor operator with the skip operation.

The Dumptor, after being loaded, is driven over a short permanent wooden trestle and directly onto the top of the stockpile. After the width of the pile has been established, each load of ore is dumped over the end of the pile and the pile is built away from the headframe. In building the pile, which in one case is 40 ft high, the operator piles all the ore on the left side of the pile during the daylight hours of work. Fig. 4 shows a Dumptor discharging ore at the edge of a stockpile. During the night shift,

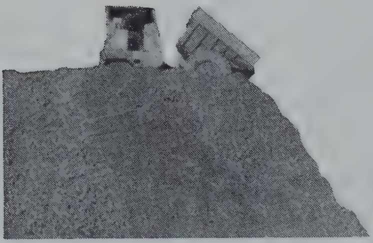

Fig. 4—A dumptor discharges ore at the edge of a stockpile.

the right side of the pile is built. This is done because the left side of the Dumptor is the blind side, and, during night hours, the left edge of the pile can thus be avoided. Working from left to right allows the left wheel on the blind side of the Dumptor to rest on the last material dumped and the distance to the edge of the pile can be estimated where the distance is shortest. The operator stops the Dumptor 3 ft from the edge of the pile and is instructed to use the spare Dumptor at the first signs of any mechanical trouble.

Maintenance work required with this system is relatively low. The top of the pile is occasionally bulldozed to smooth the surface or to remove snow. The roadway, once dozed smooth, freezes solid and requires no further work. The trestle requires new planking about once every 2 years and is completely rebuilt every 5 years because the legs within the stockpile decay. A spare Dumptor usually is used for emergency, and one Dumptor engine is torn down and rebuilt each year. The tires are replaced or repaired, if necessary, during the summer. Conversion from stocking to direct railroad car loading is simple. A portable chute is placed under the storage pocket in place of the Dumptor, and this chute feeds the crushed ore directly into car-loading pockets underneath.

Mining Operation

In underground operations, practices have changed somewhat in recent years. These changes include long hole drilling, underground blasting, main drifting and shaft sinking, and mine supporting.

Table VIII. Results of Tests with Ring Blasts

Average burden ....10 to 20 ft
Interval between subs 6th to 7th........................ 30 ft

Calculated tons from blast:
11x75x30 ft plus 11x38x26 ft = 35,600 cu ft
Less area previously sliced out [(17x10x12)2 = 4100 cu ft]
35,600 less 4100 = 31,500 cu ft from blast
31,500 divided by 12 = 2625 tons or 525 cars

plus 5°, 20°, 40°, 60°, 90°

Number of holes........................ 14
Total length of holes........................ 358 ft
Powder used—40 pct Special Gelatin, 232 sticks or 696 lbs, plus 500 ft of Primacord

Tons broken per lb of powder—(2625 ÷ 696) .... 3.77 tons
Tons broken per ft of hole—(2625 ÷ 358) ....7.33 tons

Condition of dirt blasted—Mostly big chunks 20 to 30 ft long
Breast of stope—Blast broke to holes

*Time required in drilling holes—Eight shifts, two men per shift
Average amount of hole drilled per shift........................45 ft
Drilling Equipment—Post-mounted, water liner for eight holes
Jackhammer and leg for four holes
Stoper for two holes

Large couplings used—None broken
No. 2—Drilled 71 ft when rod broke and spoiled thread
No. 6—Drilled 24 ft then lost in hole because of broken rod
No. 4—Drilled 232 ft and still good

New bits drilled 30 to 60 ft before dulling
Used and reground bits drilled 6 to 30 ft
Time required for loading holes........................3 hr, 20 min
* (Includes all delays and time for other pertinent work)

One operation uses long hole blasting to break material in the stopes. Ore in the area of this mine is softer than the average of the ore on the Range. This fact, along with this operator's ability to control the system, makes possible the use of long hole blasting. Several operators, because of the black slate hazards and because of the secondary blasting in hard ground, did not adopt this system after several comprehensive tests. Long holes are used to remove pillars at other operations, but ring blasting as the predominant method of blasting ore is carried on only at this one property. Results of several tests with ring blasts are shown in Table VIII, and Fig. 5 illustrates long hole drilling using post mounted drifter, as described in Table VIII.

Table IX shows data for long hole drilling using millisecond delay blasting caps, and Fig. 6 illustrates these data.

There appears to be a trend on the Menominee Range toward electric blasting. This fact is shown by the 100 pct use of electric caps by one operator and the partial use of electric caps by practically every operator. In the case of the 100 pct electric blasting, the reason behind its use is the very wet condition in this particular mine. Another operator is experimenting with the thought in mind of going to 100 pct electric blasting because of the recommendation of the safety dept. to make this change.

One item introduced recently in connection with blasting is the millisecond delay. These delays have been used successfully in long hole ring blasting to reduce concussion and improve material size. They also have been used successfully in main level drifting and have shown a powder saving of as much as 20 pct. Some operators complain that, because of the millisecond delays, their drift cuts have not broken completely. However, these delays have decided advantages and gradually will be used in more and more applications.

Another change in blasting practice came about by the introduction of large powder. This powder, 2x16 in., is used by most operators in one or another phase of their operations. One large operator introduced this method of blasting in the stopes, and most of the operators do some of it. The large powder increases the burden on the drill holes, improves

Table IX. Long Hole Drilling Using Millisecond Delay Caps

Theoretical tons from blast—

$$\frac{(9.5\text{x}75\text{x}30\text{ ft} - 9.5\text{x}25\text{x}8\text{ ft})}{12\text{ cu ft per ton}} = 1624\text{ tons or }325\text{ cars}$$

Number of holes........................ 10
Length of holes........................257 ft
Powder used (45 pct Gelex No. 2)............157 sticks (314 lb)

Tons broken per lb of powder ....5.17 tons
Tons broken per ft of hole ....6.32 tons

Conditions of dirt after blasting—Chunks 2 to 8 ft, hard to break on grizzly

Breast of stope after blast—Broke to holes. Bottom right hole apparently didn't break. By measurement, all three ring blasts left 4 or 5 ft against east pillar.

Difficulty in drilling this round of hole encountered because of breaking drill steel at threaded ends.

Electric blasting with Millisecond Delay—Two caps in each primer prevents cutoffs.

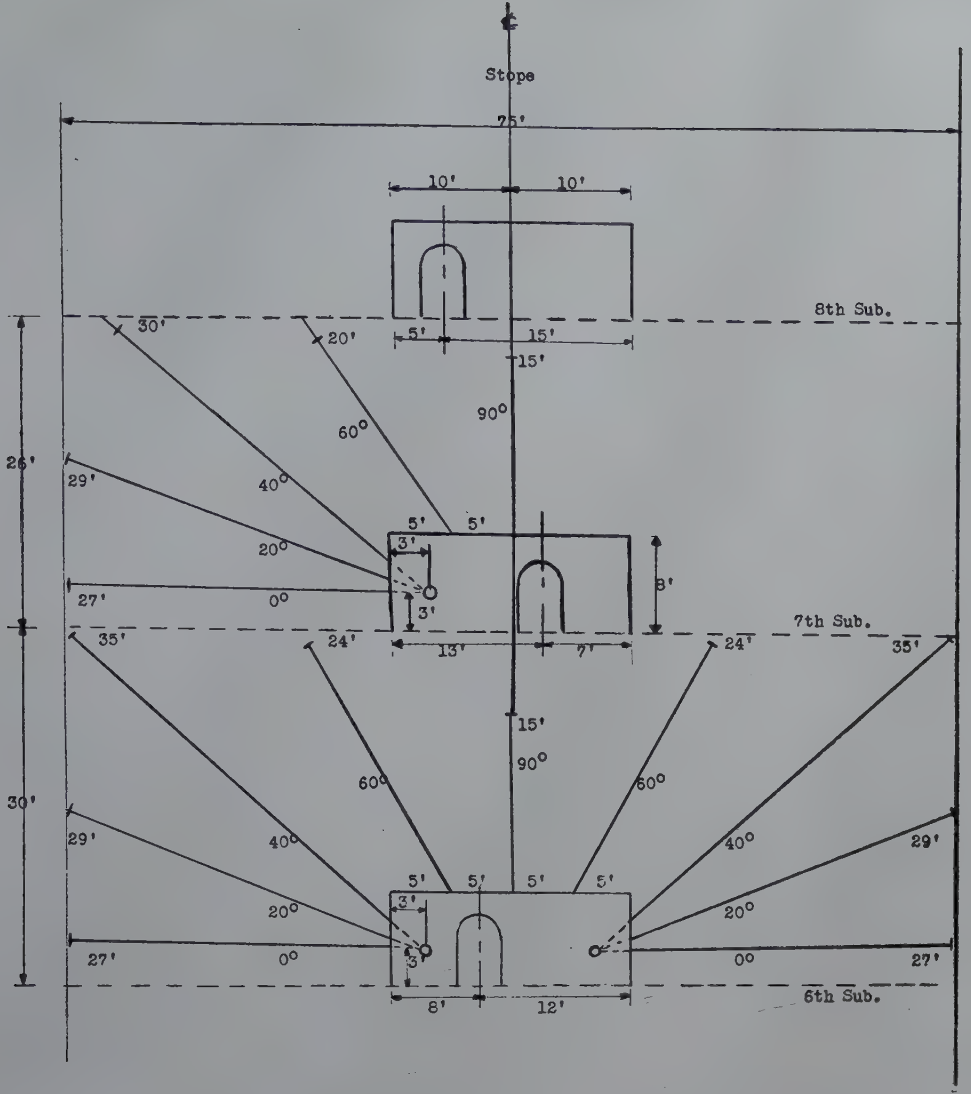

Fig. 5—Long hole drilling using post mounted drifter.

breakage, and increases the tons of material broken per pound of powder used.

Some recent methods in main drifting, which are different than in past years, are: 1—Use of the long feed automatic jumbos, 2—tungsten carbide bits, 3—large powder, 4—millisecond delays, 5—conveyor belts, and 6—improved ventilation. The shaft sinking operations have been improved by mechanical loading. Most sinking on this range in recent years has been between levels for a distance of 200 ft or less.

There are several methods of mine support employed on the Menominee Range. Included are untreated timber sets, treated timber sets, steel sets, and, in recent months on an experimental scale, roof bolts. One large operator installed a timber treating plant and treats all the underground timber. Figures prove savings over a long period of time.

Under dry conditions, steel sets are sometimes used. These sets are painted with acid-resisting paint. They have a lower installation cost than timber sets, but cost three times as much as timber. Experiments are being conducted to replace with steel hat-sections the wooden poles used over the top of these steel sets.

One operator has used roof bolts in a drift and to support ground that is not particularly heavy in other areas. Roof bolting doubtlessly will have limited use, but more tests will be necessary.

Water problems on this range are surface water and underground acid water. By the use of churn drill test holes, one operator determined that an underground lake existed over the working places of his mine. As ordinary ground acts as the shore line and bottom of a surface lake, so a ledge acts as a shore line and bottom of an underground lake. These test holes that gave the water elevations of this lake also indicated the elevation of the ledge material. From this information, a ledge contour map was made and the shore line of the lake was interpreted. It was shown to be a subterranean ravine running northwest and southeast over the orebodies and working places of the mine.

The amount of water that will enter a mine depends on two things, the ease with which it may seep through the ground and the head of water that causes this seepage. Nothing can be done to change the character of the ground, but the head of water over this ground can be controlled by the use of deep well pumps.

There were two methods employed for locating deep well pumps at this mine. The first was to place a number of deep wells to ledge across the southeast end and another group across the northwest end of the ravine. It was expected that by this system, the inflow into the area between the groups of pumps would be stopped and that the water elevation

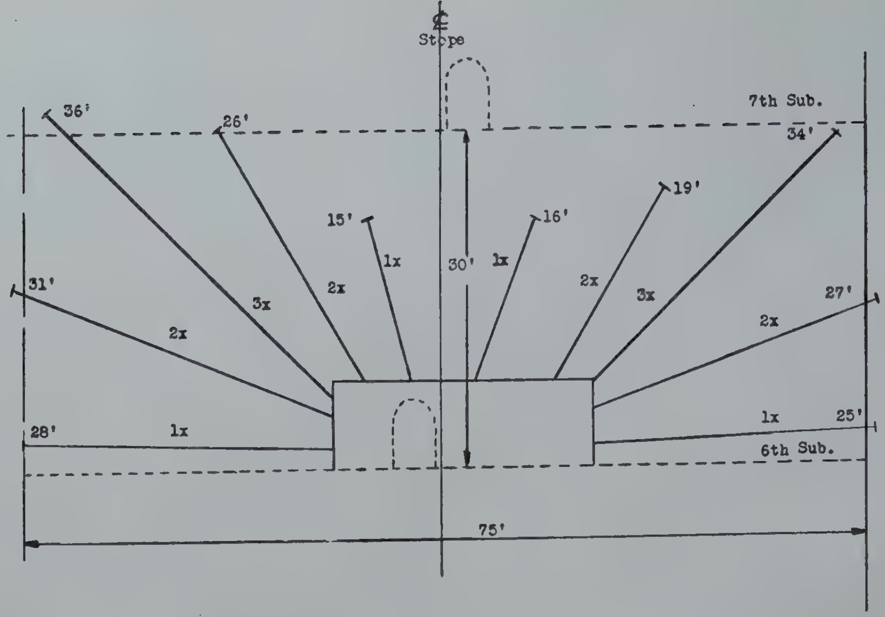

Fig. 6—Long hole drilling using millisecond delay caps.

eventually would be lowered. This method has been successful.

The second method is based on the fact that water will seep from a higher to a lower elevation in the iron formation on this range. On this theory, deep wells were sunk to the ledge as near to the iron formation as possible and directly over the iron formation in most cases. This technique reduced seepage into the working places.

Some of the results of the deep well pumping to date are: 1—All places where shafts enter ledge are presently above water except one, which is 18 ft below water. 2—Stope filling raises can now be made to surface without a water seal at the ledge, since they will be above water. Likewise a great deal more dirt to be used for stope filling is now available because it is above water. 3—The depth of water in this underground ravine has been reduced about 60 ft and therefore the head has been reduced. 4—Even though the total water pumped by both mine pumps and deep well pumps is greater than that which would be pumped with mine pumps alone, the horsepower requirements are less since the head requirements with the deep well pumps are about eight times less. 5—Deep well pumps seem to eliminate or reduce the peak pumping normally required because of spring run off.

From experience with deep wells on this range, it appears they are effective in controlling surface water above and in the area of ore bodies. There are two important considerations regarding the location of pumps: The contours of areas to be pumped, and the permeability of ground at pump locations.

Deep well pumping has been successful on the Menominee Range, and operators on other ranges have employed this same system to control the surface water above orebodies.

Quite a number of mines on the Menominee Range have acid in the underground water. One operator has had this problem to contend with for some time. The water contains sulphuric acid. The strength of the acid in this water varies from very weak on some levels to very strong on other levels. The life of a 4-in. iron pipe in some of the stronger water was less than 10 days. This operator attempted to mix the strong acid water with the weak acid water before pumping. However, at one main pumphouse the water is 0.8 pct sulphuric acid. Lime was added to the water, but this was not successful because of the quantity required.

This operator found it satisfactory to install pumps and piping of such material that the acid has little effect. Plunger pumps have been protected by spraying stainless steel on the poles, cement lining the pots, and using stainless valves and throat bushings. Porcelain poles have been used satisfactorily. However, sometimes chipping has been excessive and, because the cost of these poles is high, they are not used to any great extent. Centrifugal pumps have been protected by using stainless steel for casings and internal parts, and by using acid resistant bronze. The acid resistant bronze does not, however, hold up as well as the stainless steel.

Pump columns and suctions have been protected by two methods: 1—*Concrete lined pipe*—This method is satisfactory. However, it has its disadvantages, such as a reduction of about ¾ in. in pipe size plus the necessity of chipping to keep the pipe clean after installation. 2—*Rubber lined pipe*—This is satisfactory but initial cost is high. It seems as though no chipping of the line for the purpose of cleaning is necessary after installation.

Valves of acid-resistant bronze, stainless steel, and cement lining have been used. The stainless steel has proved best but is most expensive. Rubber lined stainless steel trim valves can be obtained, but none have been tried. However, they should be satisfactory. A new type plastic pipe has been tried for low pressures and for suction lines. This will likely prove satisfactory.

Automatic pumping at one operation would be desirable because the daily inflow of water to the mine can be stored and pumped to surface from one level, the pumping being done between shifts to avoid affecting the electrical peak. This former arrangement necessitated sending a pumpman underground between shifts. The system as installed consists of three parts, the priming circuit, the starting circuit, and the alarm circuit. It is diagrammed in Fig. 7.

The priming circuit consists of a priming tank,

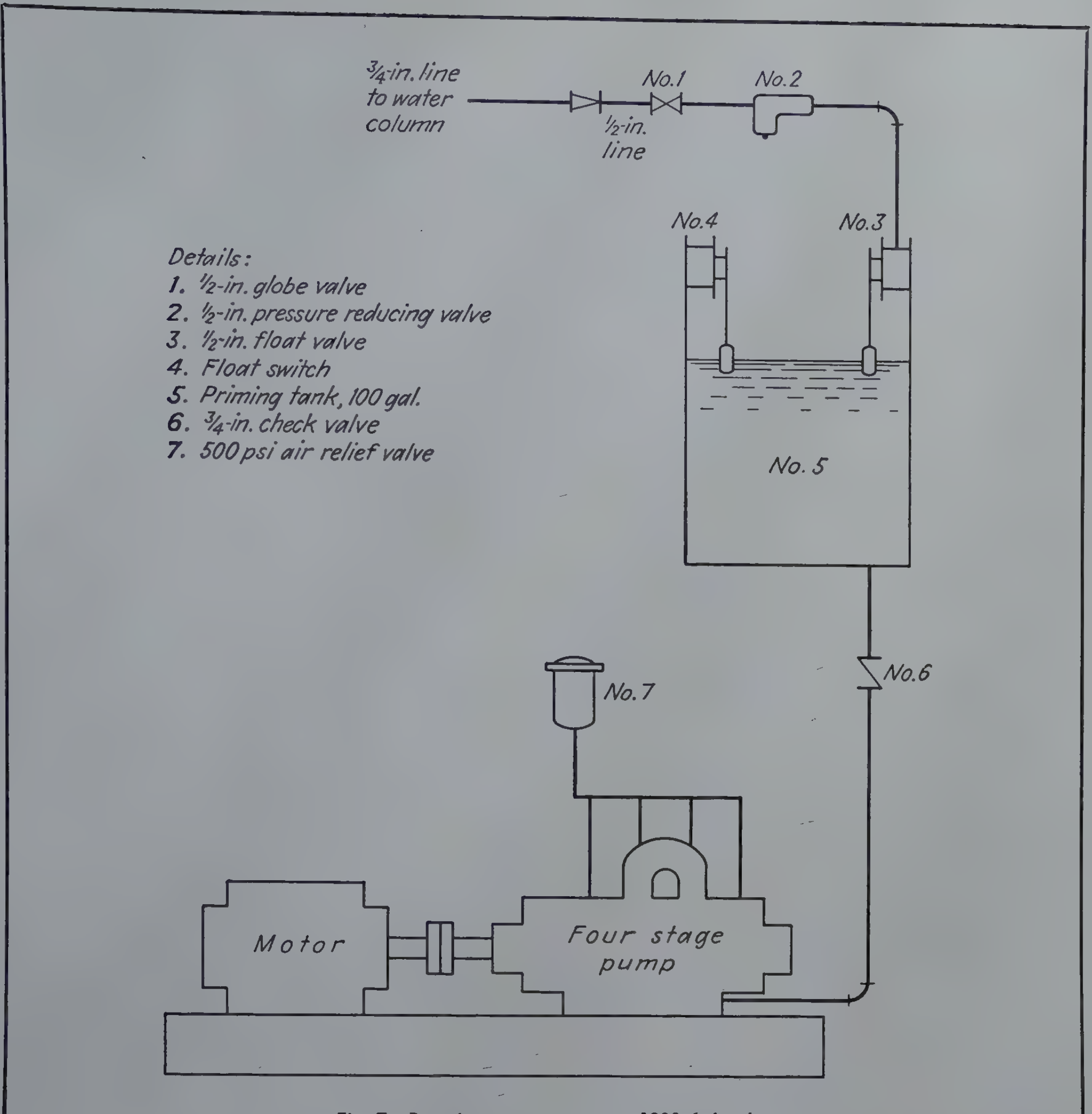

Fig. 7—Pumping arrangement at 1000-ft level.

No. 5 in Fig. 7, connected to a constant source of water through a flow-control valve, No. 3. This valve maintains the tank water level. The tank is mounted above the pump, and water flows to the pump by gravity to keep it primed. The air is bled from the pump casing by an air vent trap, No. 7 in Fig. 7.

The starting circuit consists of a magnetic full voltage starter for the 250-hp motor with switches in series to start and stop the motor. The switches are: A time clock to allow only off-peak operation, a float switch to assure sufficient water in the sump, a timing relay and pressure switch to make sure the pump builds up pressure in a designated time, and a float switch (No. 4 in Fig. 7) in the priming tank to check on the water level in it.

The alarm circuit, installed in the engine house where there is constant attendance, consists of four lights and an alarm. A green light indicates low water level in the sump, an amber light indicates sufficient water for pumping but *on-peak-time,* a white light indicates the pump is running satisfactorily, and a red light indicates that the time clock and float call for pump operation but something is wrong. Along with the red light, a horn sounds. This can be turned off manually.

Winter ventilation, stope filling, sulphur fires, and safety and management are also subjects of interest and items where developments have taken place. Mine ventilation has become more important in the past few years. This probably because mines have become larger, and because more thought has been given to the importance of fresh air at the working place. Most plants on this range steam heat the air for underground, a common method. There is, however, a trend to utilize warm air furnaces to heat the air for underground.

In one system of this nature, a fan forces air through a warm air furnace where the air is heated and then is sent underground. Bonnet temperature of the furnace controls the stoker or oil burner. The amount of air passed through the furnace is governed by thermostatic controlled dampers that permit a portion or all of the air to go through the furnace, or allow it to bypass the furnace. Thus, a constant temperature of about 36°F is maintained. This unit has been used where up to 10,000 cfm of air is required.

At another installation, where a maximum capacity of 80,000 cfm is used, a warm air furnace and two fans are used. One fan of 50,000 cfm takes in fresh air and discharges this air through the warm air furnace into a mixing chamber where the fresh air from outside also can be drawn in through back-draft louvers. From this chamber, a large 80,000-cfm fan, under 5-in. static pressure with fluid coupling to give variable speed, draws in the fresh and heated air and discharges it down a shaft into the mine. The small 50,000-cfm fan operates continuously when the furnace is in use and is shut down when the furnace is not in use. This fan merely provides a positive pressure on the furnace so that any gases from the furnace do not enter the fresh air.

The speed of the large fan is controlled by outside air temperature through the use of the fluid coupling. This control is predetermined and set so that as the temperature drops, the capacity of the large fan is reduced and thus will not exceed the capacity of the furnace.

The stoker of the furnace is controlled by discharged air temperature entering the shaft. This gives final close control of the air temperature going down the shaft and is set about 36°F.

The principal advantage of warm air heating is that there are no steam or water lines to freeze. It is also believed that the warm air system is more efficient, since lower stack temperatures are maintained and heat loss to the boiler room is reduced.

Sulphur black slate exists in both the hanging and foot walls of mines on this range. This black slate will spontaneously ignite when it is piled more than 6 to 8 ft deep. Sometimes it is impossible to keep the black slate from coming into an open stope. When this happens, a fire is likely to start, and the only successful way known to put it out is to cut off the air to the stope. Sometimes bulkheads will accomplish this. If bulkheads cannot be used, then the fire must be smothered with dirt. This is accomplished by stope filling.

Stope filling has three main purposes. It reduces the possibility of material sloughing off an old stope and thereby causing a fire. It extinguishes a burning fire. It makes possible the recovery of pillars left in the mining operations.

This stope filling is accomplished through a system of raises and churn drill holes, 30 in. diam. Surface material is allowed to fill one stope, then the floor pillar is blasted out, and the material is allowed to drop into the stope below. Sometimes these stopes do not exist one above the other, and this material is conveyed by belt from one raise to another. After this surface material has remained in a stope for a few years, it is believed safe to return and rob the pillars. To do this a thin wall of ore is left next to the stope fill, so that this fill will not run into the pillar being mined. Several operators in the district are working toward this end.

Companies on the Menominee Range are safety conscious, as is indicated by their records. Two companies have a 100 pct goggle rule. There appears to be a trend toward more training of shift bosses and mine supervisors. This trend seems necessary because the shift boss and the captain must be prepared to make decisions on the spot before they have had an opportunity to consult with supervisors. Also, a good job by these men can prevent grievances and add to the harmonious operation of these mines.

Conclusion

A method to set forth the summation of the information in this report might be to describe an ideal property, using the methods that the operators feel are best suited to the conditions on this range, keeping in mind the trends and the progress made during recent years. An ideal mine might be as follows:

I. Drilling Methods:

A. One-use bits in soft ground.
B. Tungsten carbide bits in hard ground.
C. Carbon steel rods prepared under carefully controlled temperatures in a well equipped shop.
D. Light hand-held machines on jack legs in soft ground. Pneumatic-feed machines on pneumatic posts in hard ground and for long holes.
E. Air pressures 100 to 110 psi.

II. Transportation:

A. Large tram cars and locomotives and heavy rails to haul hard massive chunks with the crusher underground. Conveyor belts for large concentrations of ore with average chunk problems. Crusher to be on surface.
B. Dumptor or similar device for stockpiling.

III. Operations:

A. Long hole ring blasting in soft ground and in pillars. Large holes and large powder, or small holes drilled with insert bits, in hard ground.
B. Electric blasting millisecond delays.
C. Tungsten carbide bits on long feed machines on pneumatic jumbo for drifting.
D. Shaft sinking with tungsten carbide bits, and clam shell muckers.
E. Mine supports of treated timber or steel sets.

IV. Water problems:

A. Deep wells to control surface water.
B. Acid-resisting pumps and pipes for underground water.
C. Automatic pumping.

V. General:

A. Heated ventilation with hot air furnace.
B. Stope filling to control sulphur fires and recover pillars.
C. Clutch drum hoist.
D. Electric eye control for skips.
E. Low headframe with conveyor belts to carry material to crusher, storage pocket, or stockpile.
F. Consolidation of shops, dry and garages, in one building, in addition to an engine house and office building. Enclosed passageways between buildings and the collar of the shaft.

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Grinding Mills as Conditioners in Sulphide Flotation

by C. G. McLachlan

Laboratory flotation tests carried out on massive sulphide ores may not be reproduced in plant practice, and when this occurs the discrepancy can be the result of differences between laboratory and plant grinding. Results of parallel flotation tests are given to support this contention, and data show how tailing losses are affected in corresponding size zones.

ONE phase of the treatment in sulphide flotation, covered generally in a review of pulp pretreatment by S. A. Falconer,[1] is the matter of grinding mills as conditioners, a subject on which further study could be advantageous. Flotation with few exceptions is preceded by grinding, and the grinding and flotation capacity required for the treatment of an ore is based generally on laboratory or pilot mill tests. The results of such tests, it then is assumed, will be duplicated in practice, and a flowsheet is prepared on that basis.

This assumption may be too broad, because in several instances laboratory flotation tests made on massive sulphide copper ores could not be duplicated in plant operation without the introduction of an aeration step, either in closed circuit with the grindinging mills, or between grinding and flotation.[2, 3]

The differences between laboratory and plant grinding practice are considerable, and may be summarized as follows:

1—In the laboratory a relatively fine charge is ground in a batch mill. In the plant, the feed, usually considerably coarser than the ore tested in the laboratory, enters the grinding mill continuously and more often than not, oversize material in the mill discharge is retained in the grinding circuit by a classifier until the required degree of reduction has been effected.

2—The rate of grinding in the laboratory is slower than in plant practice.

3—The percentage of volume occupied by the pulp in a laboratory mill rarely approaches that of a continuous overflow type mill. This does not apply if the comparison is made with grate type mills, but in that case an even wider discrepancy between the relative rates of laboratory and mill grinding almost invariably exists.

The present trend in grinding is toward larger mills to reduce the number of operating units. This accentuates the foregoing differences. It is believed that the trend toward larger mills is sound, but that when large mills are used, metallurgists should check their flotation circuits carefully to determine whether laboratory tests are being reproduced in

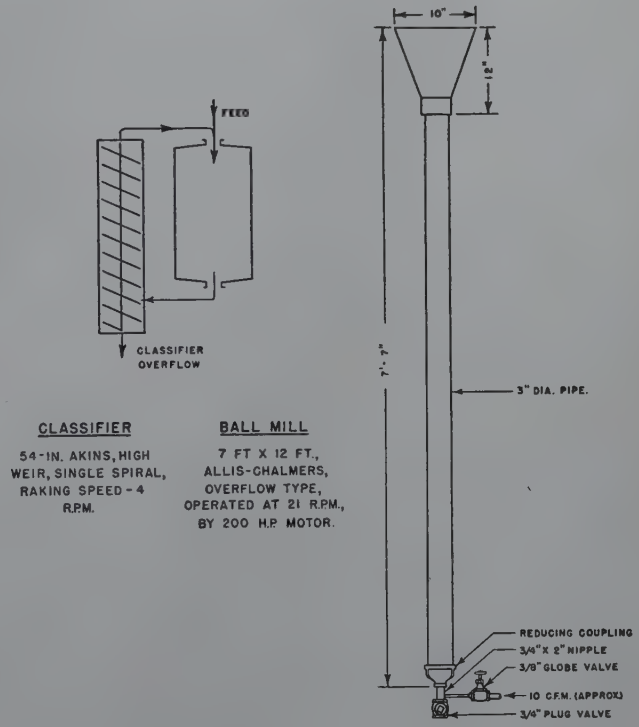
FEED
CLASSIFIER OVERFLOW
CLASSIFIER
54-IN. AKINS, HIGH WEIR, SINGLE SPIRAL, RAKING SPEED - 4 R.P.M.
BALL MILL
7 FT X 12 FT., ALLIS-CHALMERS, OVERFLOW TYPE, OPERATED AT 21 R.P.M., BY 200 H.P. MOTOR.
10"
12"
7'-7"
3" DIA. PIPE.
REDUCING COUPLING
3/4"X 2" NIPPLE
3/8" GLOBE VALVE
10 C.F.M. (APPROX)
3/4" PLUG VALVE

Fig. 1 (Left)—Grinding flowsheet used for mill tests. (Right)—Laborator aerator.

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Table I. Tests with Lime

Treatment Before Flotation	Product	Plant Grinding: Cu Assay, Pct	Plant Grinding: Cu Recovery, Pct	Laboratory Grinding: Cu Assay, Pct	Laboratory Grinding: Cu Recovery, Pct
None	Heads	1.43		1.39	
	Conc.	6.24	25.8	6.67	77.6
	Tail.	1.13		0.37	
20-min agitation laboratory flotation cell, air inlet closed	Heads	1.42		1.39	
	Conc.	6.34	27.2	6.32	66.2
	Tail.	1.10		0.55	
20-min agitation laboratory flotation cell, air inlet open	Heads	1.42		1.40	
	Conc.	10.64	54.3	12.98	85.8
	Tail.	0.70		0.22	
20-min laboratory aerator	Heads	1.42		1.39	
	Conc.	10.31	49.6	11.24	91.1
	Tail.	0.77		0.14	

Table II. Tests with Soda Ash and Cyanide

Treatment Before Flotation	Product	Plant Grinding: Cu Assay, Pct	Plant Grinding: Cu Recovery, Pct	Laboratory Grinding: Cu Assay, Pct	Laboratory Grinding: Cu Recovery, Pct
None	Heads	2.67		2.60	
	Conc.	7.93	47.4	9.44	64.2
	Tail.	1.67		1.13	
20-min agitation laboratory flotation cell, air inlet closed	Heads	2.77		2.61	
	Conc.	9.10	73.1	6.87	86.4
	Tail.	0.96		0.53	
20-min agitation laboratory flotation cell, air inlet open	Heads	Corresponding test not run		2.60	
	Conc.			10.56	82.5
	Tail.			0.57	
20-min laboratory aerator	Heads	2.77		2.61	
	Conc.	14.56	80.4	14.28	92.0
	Tail.	0.64		0.25	

Table III. Linking Tests Using Laboratory Grinding and 20-Min Aeration

Test	Product	Tests With Lime: Cu Assay, Pct	Tests With Lime: Cu Recovery, Pct	Tests With Soda Ash: Cu Assay, Pct	Tests With Soda Ash: Cu Recovery, Pct
Linking test for lime series	Heads	1.39		1.40	
	Conc.	11.24	91.1	9.94	91.9
	Tail.	0.14		0.13	
Linking test for soda ash series	Heads	2.62		2.61	
	Conc.	12.35	89.8	14.28	92.0
	Tail.	0.33		0.25	

plant practice. This usually can be done by taking a sample of classifier overflow, or flotation feed, and comparing the flotation results obtained in the laboratory with those of a duplicate test in which a sample of the mill feed is ground in the laboratory to the same degree of fineness and subjected to identical flotation treatment.

In those cases where satisfactory agreement between laboratory and mill flotation results does not exist because of grinding circuit differences, it has been found that they most frequently occur when the following types of ore are being treated: 1—Massive sulphide ores, particularly those containing pyrrhotite, 2—ores containing reducing agents such as "ous" salts, and 3—ores forming colloidal slimes when ground. This list is not intended to be complete, but it should be indicative.

The term duplication of results means not only duplication of metallurgical performance and the use of equivalent quantities of flotation reagents, but also duplication of the actual flotation periods required for treating respective pulps.

An example of the type of discrepancy referred to is shown by the results of the following series of parallel flotation tests which were carried out on laboratory, and mill ground products, with ore currently treated by the Noranda concentrator.

The detailed procedure used for these tests and other related data will be found in the appendix. A flowsheet of the mill grinding circuit employed and the laboratory aerator are shown in Fig. 1.

Table IV. Tabulation of Assays of Sized Tailings Products

Where Ground	Treatment Before Flotation	Alkali	Micron Zones: +56	−56 +40	−40 +28	−28 +20	−20 +14	−14 +10	−10	Cum.
Plant	None	Lime	1.13	1.25	0.94	0.75	0.79	1.05	1.51	1.13
Laboratory			0.46	0.15	0.11	0.11	0.13	0.22	0.84	0.37
Plant	20-min agitation laboratory flotation cell, air inlet closed	Lime	1.12	1.25	0.84	0.64	0.66	0.89	1.46	1.10
Laboratory			0.65	0.24	0.10	0.15	0.20	0.48	1.25	0.55
Plant	20-min agitation laboratory flotation cell, air inlet open	Lime	0.98	0.38	0.17	0.12	0.14	0.26	0.98	0.70
Laboratory			0.29	0.09	0.07	0.07	0.07	0.11	0.42	0.22
Plant	20-min laboratory aerator	Lime	0.98	0.40	0.24	0.18	0.21	0.37	1.20	0.77
Laboratory			0.16	0.07	0.05	0.05	0.06	0.10	0.36	0.14
Laboratory link test	20-min laboratory aerator	Soda ash	0.12	0.08	0.07	0.07	0.09	0.10	0.38	0.13
Plant	None	Soda ash	2.18	1.59	0.69	0.39	0.33	0.48	2.15	1.67
Laboratory			1.27	0.51	0.37	0.36	0.59	1.33	2.31	1.13
Plant	20-min agitation laboratory flotation cell, air inlet closed	Soda ash	1.25	0.25	0.16	0.16	0.18	0.49	2.08	0.96
Laboratory			0.58	0.12	0.11	0.12	0.17	0.46	1.47	0.53
Laboratory	Air inlet open	Soda ash	0.66	0.11	0.09	0.11	0.18	0.45	1.20	0.57
Plant	20-min laboratory aerator	Soda ash	0.96	0.20	0.15	0.11	0.15	0.21	1.03	0.64
Laboratory			0.26	0.11	0.10	0.09	0.13	0.27	0.79	0.25
Laboratory link test	20-min laboratory aerator	Lime	0.41	0.13	0.10	0.10	0.10	0.15	0.45	0.33

Table V. Flotation Tailings Infrasizing Analyses, Lime Series (Data for Table IV)

Type of Grinding	Treatment Before Flotation	Alkali	Product, Pct	Micron Zones							Total	Tailing (calc.)
				+56u	56-40	40-28	28-20	20-14	14-10	—10		
Plant	None	Lime	Weight	52.2	10.4	8.7	6.0	4.7	4.9	13.1	100.0	1.13
			Assay Cu	1.13	1.25	0.94	0.75	0.79	1.05	1.51		
			Distribution Cu	52.0	11.5	7.2	4.0	3.3	4.6	17.4	100.0	
Laboratory	None	Lime	Weight	49.8	12.5	10.4	7.0	5.2	3.9	11.2	100.0	0.37
			Assay Cu	0.46	0.15	0.11	0.11	0.13	0.22	0.84		
			Distribution Cu	60.9	5.0	3.1	2.0	1.8	2.3	24.9	100.0	
Plant	20-min agitation laboratory flotation cell, air inlet closed	Lime	Weight	49.2	10.3	9.2	6.4	5.0	4.2	15.7	100.0	1.10
			Assay Cu	1.12	1.25	0.84	0.64	0.66	0.89	1.46		
			Distribution Cu	50.2	11.8	7.0	3.7	3.0	3.4	20.9	100.0	
Laboratory	20-min agitation laboratory flotation cell, air inlet closed	Lime	Weight	52.2	11.6	9.8	6.7	4.8	4.1	10.8	100.0	0.55
			Assay Cu	0.65	0.24	0.10	0.15	0.20	0.48	1.25		
			Distribution Cu	61.8	5.1	1.8	1.8	1.7	3.1	24.7	100.0	
Plant	20-min agitation laboratory flotation cell, air inlet open	Lime	Weight	47.8	11.2	9.4	6.8	5.2	4.8	14.8	100.0	0.70
			Assay Cu	0.98	0.38	0.17	0.12	0.14	0.26	0.98		
			Distribution Cu	69.0	5.7	2.1	1.1	1.0	1.7	19.4	100.0	
Laboratory	20-min agitation laboratory flotation cell, air inlet open	Lime	Weight	52.1	12.3	10.2	6.7	4.8	4.1	9.8	100.0	0.22
			Assay Cu	0.29	0.09	0.07	0.07	0.07	0.11	0.42		
			Distribution Cu	67.8	5.0	3.2	2.1	1.5	2.0	18.4	100.0	
Plant	20-min laboratory aerator	Lime	Weight	52.3	10.3	8.8	6.2	5.0	4.3	13.1	100.0	0.77
			Assay Cu	0.98	0.40	0.24	0.18	0.21	0.37	1.20		
			Distribution Cu	66.6	5.3	2.7	1.5	1.4	2.1	20.4	100.0	
Laboratory	20-min laboratory aerator	Lime	Weight	50.3	12.5	10.4	7.0	5.1	3.8	10.9	100.0	0.14
			Assay Cu	0.16	0.07	0.05	0.05	0.06	0.10	0.36		
			Distribution Cu	55.9	6.1	3.6	2.4	2.1	2.6	27.3	100.0	
Laboratory link test	20-min laboratory aerator	Soda ash	Weight	47.9	12.4	10.6	7.3	5.6	5.1	11.1	100.0	0.13
			Assay Cu	0.12	0.08	0.07	0.07	0.09	0.10	0.38		
			Distribution Cu	43.5	7.5	5.6	3.9	3.8	3.8	31.9	100.0	

Table I shows the results obtained in a series of tests in which lime was added to the grinding mill to provide protective alkalinity.

Examination of the figures given in this table shows that a very considerable difference exists between the flotation results obtained on the mill-ground and the laboratory-ground pulps; also, that these variations are influenced by the method used for treating the pulp after grinding and before flotation.

At Noranda these variations can be reduced by using soda ash and cyanide in place of lime. Confirmation of this statement will be found by comparing the recovery figures given in Table II with those of Table I.

In spite of the reduction in the discrepancy shown by the results given in Table II, a serious difference still exists in favor of the laboratory. In plant practice it has been found that if a soda ash circuit is used, satisfactory correlation between the mill and the best laboratory tests can be obtained by operating aerators in closed circuit with the grinding mill and bringing back a fair circulating load of middling, which is reaerated before being refloated. Xanthate consumption in the mill is, however, somewhat higher than indicated by the laboratory tests, and certain auxiliary circuits, which are employed to recover finely interlocked gold, also recover some copper. Actually, plant performance for the period within which the tests listed in Table II were car-

Table VI. Flotation Tailings Infrasizing Analyses, Soda Ash Series (Data for Table IV)

Type of Grinding	Treatment Before Flotation	Alkali	Product, Pct	Micron Zones							Total	Tailing (calc.)
				+56u	56-40	40-28	28-20	20-14	14-10	—10		
Plant	None	Soda ash	Weight	46.9	10.8	9.8	6.6	5.3	4.7	15.9	100.0	1.67
			Assay Cu	2.18	1.59	0.69	0.39	0.33	0.48	2.15		
			Distribution Cu	61.3	10.3	4.0	1.5	1.1	1.3	20.5	100.0	
Laboratory	None	Soda ash	Weight	65.7	8.3	7.2	5.0	4.4	3.5	5.9	100.0	1.13
			Assay Cu	1.27	0.51	0.37	0.36	0.59	1.33	2.31		
			Distribution Cu	73.8	3.7	2.4	1.6	2.3	4.1	12.1	100.0	
Plant	20-min agitation laboratory flotation cell, air inlet closed	Soda ash	Weight	41.7	11.0	10.2	7.5	6.3	7.2	16.1	100.0	0.96
			Assay Cu	1.25	0.25	0.16	0.16	0.18	0.49	2.08		
			Distribution Cu	54.3	2.9	1.7	1.2	1.2	3.7	35.0	100.0	
Laboratory	20-min agitation laboratory flotation cell, air inlet closed	Soda ash	Weight	65.8	7.9	6.8	4.8	3.9	3.7	7.1	100.0	0.53
			Assay Cu	0.58	0.12	0.11	0.12	0.17	0.46	1.47		
			Distribution Cu	71.7	1.8	1.4	1.1	1.2	3.2	19.6	100.0	
Plant	20-min agitation laboratory flotation cell, air inlet open	Soda ash	Weight	Corresponding test not run								
			Assay Cu									
			Distribution Cu									
Laboratory	20-min agitation laboratory flotation cell, air inlet open	Soda ash	Weight	65.0	7.8	7.1	5.4	4.1	2.5	8.1	100.0	0.57
			Assay Cu	0.66	0.11	0.09	0.11	0.18	0.45	1.20		
			Distribution Cu	75.8	1.5	1.1	1.1	1.3	2.0	17.2	100.0	
Plant	20-min laboratory aeration	Soda ash	Weight	41.0	11.5	10.8	7.4	6.2	6.3	16.8	100.0	0.64
			Assay Cu	0.96	0.20	0.15	0.11	0.15	0.21	1.03		
			Distribution Cu	61.9	3.6	2.5	1.3	1.5	2.1	27.1	100.0	
Laboratory	20-min laboratory aeration	Soda ash	Weight	63.3	8.6	7.7	5.6	4.9	4.4	5.5	100.0	0.25
			Assay Cu	0.26	0.11	0.10	0.09	0.13	0.27	0.79		
			Distribution Cu	66.2	3.8	3.1	2.0	2.6	4.8	17.5	100.0	
Laboratory (link test)	20-min laboratory aeration	Lime	Weight	64.5	8.5	7.3	4.8	3.7	2.9	8.3	100.0	0.33
			Assay Cu	0.41	0.13	0.10	0.10	0.10	0.15	0.45		
			Distribution Cu	79.5	3.3	2.2	1.4	1.1	1.3	11.2	100.0	

ried out, showed a copper recovery of 95.9 pct and a ratio of concentration of 6.5:1. Repeated attempts have been made in the mill to duplicate the best laboratory results with lime but these have always failed.

The copper content of the two lots of ore tested in Tables I and II varied considerably because the two series were not run on the same day. To provide for this eventuality and to enable a better comparison between each series, two laboratory linking tests were run, one with soda ash and cyanide plus aeration on the sample of ore used for the lime tests, and the other with lime plus aeration on the sample used for the soda ash tests. Results for both tests are given in Table III, where for purposes of comparison, those for the corresponding opposite tests already recorded in Table I and II are shown.

The tailings for the tests covered by Tables I, II, and III were infrasized,[4] so that the copper assays and weights in the different size zones could be determined and compared. These data will be found tabulated in detail in Tables V and VI, but the assays of the sized products have been summarized for convenience and are listed in Table IV.

In comparing these sizing data, consideration must be given to the fact that the tailings for the soda ash series were made on ore averaging 2.65 pct copper, while the ore for the lime series contained only 1.40 pct copper.

It should be noted that if the plant-ground and laboratory-ground products are considered as comprising two separate groups, the tests in which the lowest tailings were made in each group were also those in which relatively high grade concentrates were produced; *also, that the lower the overall tailing, the flatter the assay curve.* These points are of interest because Noranda ore is relatively fine grained, and yet the figures show that the conditions established prior to flotation are much more important in determining the grade of concentrate and the copper recovery than any degree of interlocking which existed within the grinding limits used.

Acknowledgment

The writer wishes to thank Noranda Mines, Ltd., for granting permission to present this paper and to acknowledge the work done by H. L. Ames, Concentrator Superintendent at Noranda, and his staff in carrying out the mill tests. Particular mention should also be given to the part played by the Noranda assay office and the care with which C. Barnett and his assistants in the mill laboratory carried out the various flotation and sizing tests.

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Appendix

Mineralogical Content of the Ore Tested

	Lime Series, Pct	Soda Ash Series, Pct
Chalcopyrite	4.1	7.8
Pyrite	23.4	24.0
Pyrrhotite	58.2	55.7
Insol	10.1	11.2

Specific Gravity of Ore, 4.1

Screen Analysis of Ball Mill Feed

Mesh	Wt, Pct	Mesh	Wt, Pct
+0.525 in.	8.7	— 20 + 28	3.1
—0.525 + 0.371	12.3	— 28 + 35	2.7
—0.371 + 3	12.4	— 35 + 48	3.3
— 3 + 4	11.7	— 48 + 65	3.2
— 4 + 6	8.1	— 65 + 100	3.4
— 6 + 8	7.2	—100 + 150	2.9
— 8 + 10	5.3	—150 + 200	1.8
— 10 + 14	4.4	—200	5.8
— 14 + 20	3.7	**Total**	100.0

Plant Grinding Circuit

Mill Operated In Closed Circuit With Akins Classifier

Tonnage:	12 tons per hr
Ball size:	maximum diameter 4 in.
Pulp density:	mill discharge 77 to 78 pct solids classifier overflow 45 to 46 pct solids.

Laboratory Grinding Data

Rod Mill:	8x15 in. (inside measurements)	Size of feed:	ore crushed to —4-mesh
Speed:	72 rpm	Pulp charge:	2000 g ore, 570 cc water
Rod charge:	25 rods, 1 to ⅝ in. diam total weight, 50 lb	Grinding period:	lime series 13 min soda ash series 8 min

Screen Analyses

	Akins Classifier Overflow		Laboratory Mill Discharge	
	Lime Series	Soda Ash Series	Lime Series	Soda Ash Series
Mesh	Wt, Pct	Wt, Pct	Wt, Pct	Wt, Pct
+65	3.0	3.6	0.1	3.2
— 65+100	10.8	7.9	2.3	15.8
—100+150	14.9	13.6	13.9	21.4
—150+200	11.1	12.3	16.6	11.7
—200	60.2	62.6	67.1	47.9

Test Procedures for Laboratory Flotation of Mill-Ground Ore

1—Reagents added to head of mill:
  a—Soda ash series: soda ash, 4.0 lb per ton
    sodium cyanide, 0.06 lb per ton
  b—Lime series: calcium hydrate, 5.0 lb per ton
    (65 pct available CaO)

2—a—Tests with no treatment before flotation: Collector (0.1 lb per ton potassium amyl xanthate) added, and sample transferred to 2000-g Denver laboratory flotation machine. Pine oil (3 drops) added and pulp diluted to 25 pct solids. Floated for 5 min.

  b—Tests using agitation with and without air before flotation: Collector added and sample transferred to Denver flotation machine. Pulp agitated 20 min at 40 pct solids, pine oil added and pulp diluted to 25 pct solids. Floated for 5 min.
  Note: When agitating without air, the speed of the flotation machine was reduced to keep air from being beaten into the pulp.

  c—Tests with aeration before flotation: Collector added and pulp transferred to laboratory aerator. Pulp aerated 20 min at 40 pct solids. Aerated pulp transferred to Denver flotation machine, pine oil added and pulp diluted to 25 pct solids. Floated for 5 min.

Procedure for Laboratory-Ground Ore, Flotation Tests

1—Reagents to laboratory rod mill.
  Test series with soda ash: soda ash, 4.0 lb per ton
    sodium cyanide, 0.06 lb per ton
  Test series with lime: lime, 5.0 lb per ton
  (See also laboratory grinding data)
2—Treatment before flotation as described in a, b and c.
  Note:
  The alkalinity dropped during agitation or aeration but in all cases the pulp had a pH of approximately 8, or higher during flotation.

Geophysics on the Pennsylvania Turnpike

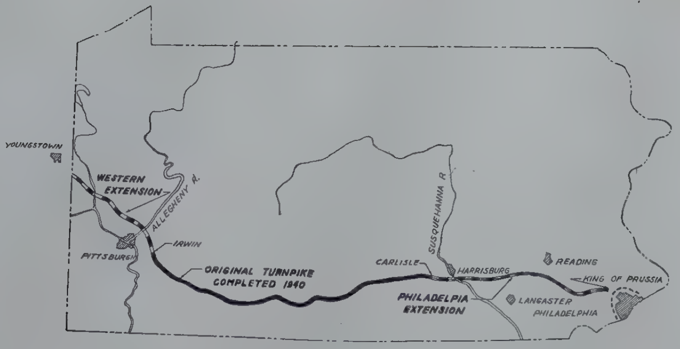

Fig. 1—Pennsylvania Turnpike system, Pennsylvania.

by H. LeRoy Scharon and A. B. Cleaves

The electrical resistivity method was utilized at 245 road cuts and structure sites over the 166 miles of the Philadelphia and Western Extensions of the Pennsylvania Turnpike System. Varying geologic structures and rock types afforded an excellent academic study of the method as well as a practical application to modern highway engineering.

THE application of geophysical techniques in the engineering surveys of the Philadelphia and Western Extensions of the Pennsylvania Turnpike System is unique inasmuch as this is the first time, (to the authors' knowledge), that such studies have been employed on a large scale on a major highway system.

These investigations were confined to specific areas along the center line of the proposed Philadelphia Extension between Carlisle and King of Prussia, and the Western Extension between Irwin and the State line near Youngstown, Ohio, as shown in Fig. 1.

Along the 100 miles of the Philadelphia Extension, the geophysical surveys were confined to 137 individual cuts and comprised a total length of 37.4 miles of line. The Western Extension, 66 miles in length, has 108 individual cuts. The accumulated length covered by electrical resistivity studies was 17.3 miles. The depth of the cuts varied from a minimum of 10 ft to a maximum of 62 ft.

The geophysical investigations were made for the purpose of determining depths to bedrock in the 245 proposed cuts and at the sites of various structures. The data also served as a basis for geological interpretations of the condition of bedrock, contact zones, rock types, water table, and other factors significant in highway construction.

The geophysical method of investigation employed the use of the electrical resistivity principle using the Gish-Rooney type electrical resistivity meter and the Wenner electrode configuration.* Apparent resistivities were determined for constant intervals to various depths and were correlated with surface geological observations and subsurface geological conditions revealed by key diamond drill holes situated within a given cut interval. From these correlations, depths to bedrock were postulated with notations on other geological features that might be pertinent to modern highway construction.

Geology

The bedrock underlying the areas of the extensions of the Turnpike afford an excellent academic study of the electrical resistivity depth method as well as a practical application to modern highway engineering.

The rocks encountered on the Philadelphia Extension vary in age from pre-Cambrian to Triassic and Quaternary, whereas in the Western Extension, bedrock is confined to relatively flat-lying strata in the Allegheny, Conemaugh, and Monongahela formations of the Upper Paleozoic, Pennsylvanian. Glacial and Recent stream, valley, and terrace deposits are common.

Philadelphia Extension

The oldest bedrock encountered in this extension are igneous and metamorphic. These, encountered chiefly in Chester County, consist of the strongly

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* F. Wenner, A Method of Measuring Earth Resistivity. U. S. Bureau of Standards *Scientific Paper* 258 (1915).

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contorted, banded Pickering gneiss which is in part graphitic and is locally injected by granite, quartz monzonite, diorite, gabbro and serpentine. Inclusions of the Franklin limestone are present. In general, the regolith is well developed over these rocks and swampy ground conditions are present in the area underlain by them.

The CambrianChickies quartzite is a ridge builder in this area. Along the Susquehanna River, downstream from Middletown, this very resistant rock stands as a cliff. However, in other areas, the cementing material between the individual sand grains has been removed by deep weathering. Consequently, although the topographic expression of a resistant ridge-building rock remains, solid rock in the true sense is lacking.

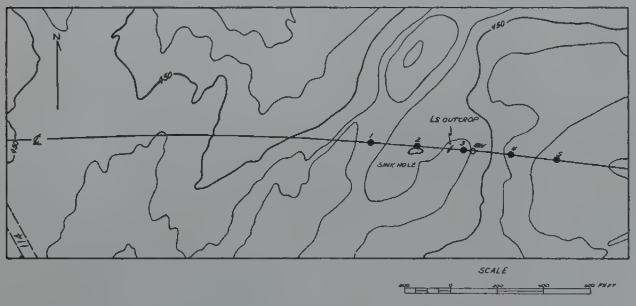
Cambrian and Ordovician limestones occur in Berks and Chester County near the eastern end of the extension and in Cumberland County near Middlesex. These limestones are strongly folded and fractured. Individual strata in the limestones vary greatly in solubility so that in a sequence of nearly vertical beds one stratum may crop at the surface of the ground whereas an adjacent stratum may be buried 30 or 40 ft in its own residium. Sinkholes in these strata are common and usually follow beds of high solubility.

Ordovician shales, Martinsburg in Cumberland County, and Cocalico in Lancaster County, show as brownish weathered, thin-bedded and strongly fractured and folded beds with a thin cover.

The most abundant rock throughout the Philadelphia Extension belong in the Triassic and consist of red clay-shales and interbedded, poorly cemented and arkosic-like sandstones and fine-grained conglomerates.

Diabasic sills in the Triassic vary considerably in grain size, and weathering characteristics. Diabase may outcrop in one spot and be weathered to great depths a few feet away. Where deeply weathered, the diabase may be said to be a relic soil. Where the sills are multiple and vary in grain size, one part of the sill may have a high moisture content and that part immediately adjacent may have a low moisture content. Commonly, within these relic soils unreduced remnants may remain as reasonably fresh rock.

Occasionally, along the northern boundary of the Triassic red sediments, faults between the Triassic and Paleozoic (Cambro-Ordovician) rock are encountered. Some have extensive development of gouge whereas others do not. The fault contacts were never observed in exposed sections but were drilled through on several occasions.



Western Extension

The rock strata in the area of this extension are of Pennsylvanian age and are relatively flat-lying. Except in glacial areas and along major stream courses the overburden is thin. The strata consist chiefly of micaceous sandstones, argillaceous sandstones, arenaceous shales, clay-shales, thin limestones, clays and coal beds. It is common to encounter firm sandstone layers underlain by clay-shales so weathered as to be essentially clays. In many sections the strata overlying mined-out areas have suffered subsidence with consequent fracturing and other disturbances.

Glacial outwash, terrace gravels, and recent sands occur along the margins of the Allegheny and Beaver Rivers and choke the valleys west of the Beaver River. The Allegheny River valley is filled with silt, sand and gravels up to 80 ft thick. On one island in the river on the proposed centerline, 66 ft of sands, gravels, silts and clays rest on a shale bedrock.

Examples of the application of the electrical resistivity measurements under these varying geological conditions consist of three on the Philadelphia extension and two on the Western Extension. These are: 1—Ordovician limestones, 2—Upper Triassic shales and sandstones, 3—Upper Triassic igneous rocks, 4—Pennsylvanian sandstones and shales, and 5—recent gravels and Pennsylvanian shale.

Geophysics

The Gish-Rooney type electrical resistivity meter and the Wenner configuration of electrode spacings were used in the determination of apparent resistivities to various depths at the sites of 245 proposed cuts and structures.

In the proposed cuts a total of 2347 separate station centers were located at intervals of 100 or 200 ft along the centerline and measured to depths varying from 18 to 180 ft. The measuring intervals varied from 1½ to 3 to 6 ft. The accuracy of interpretation is a function of the measuring interval. As nearly as possible, the line of electrode configuration was parallel to the centerline.

It is impracticable to discuss the results in all of the cuts; therefore, selected examples have been

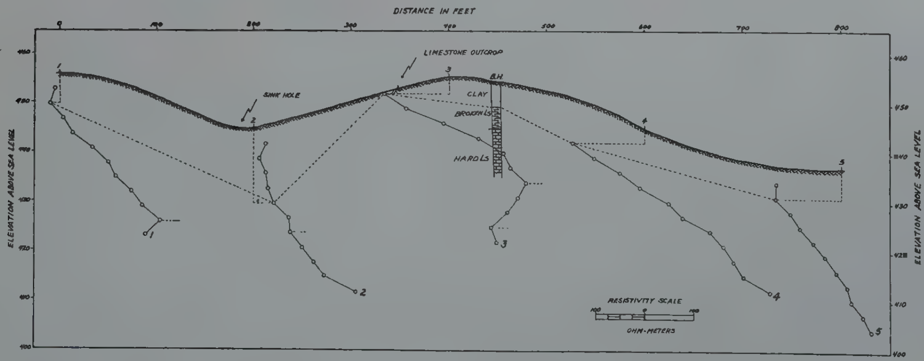


Fig. 2—Cut near Mechanicsburg, Cumberland County, Philadelphia Extension, Pennsylvania Turnpike.

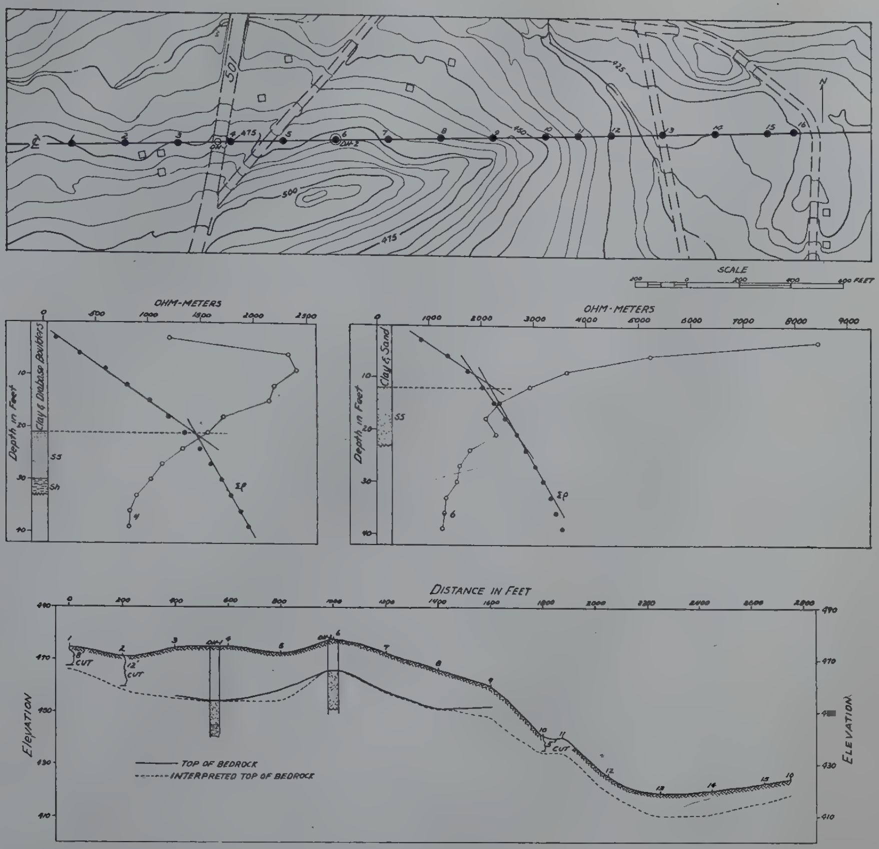


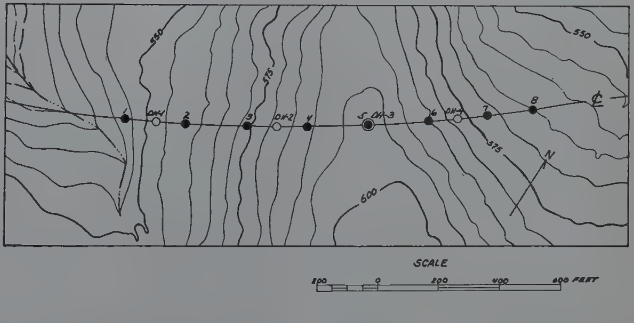
Fig. 3—Cut in vicinity of Route 501, Lancaster County, Philadelphia Extension, Pennsylvania Turnpike.

chosen, each representing different geological conditions and rock types to demonstrate the applications and results that may be expected under similar conditions.

Cambro-Ordovician Limestone

In a proposed cut near Mechanicsburg, Cumberland County, Philadelphia Extension, limestone outcrops and sinkholes occur in a matter of tens of feet from each other along the centerline, as shown in Fig. 2. These limestones are Cambro-Ordovician in age. The strata strike N45E, witnessed by trend of topography just west of station 1, and dip 29° to 50° southeast. In this particular cut-area, no ground water was reported.

Bedrock was determined in one borehole at a depth of 5 ft with sound rock at 9½ ft. An outcrop was recorded 100 ft west of the borehole and a sinkhole 150 ft further west. Five electrical resistivity stations were established at 200-ft intervals. Station 3, 50 ft west of the borehole, indicates bedrock depth at less than 3 ft (close to outcrop) whereas at stations 4 and 5 bedrock occurs at depths of 3 and 6 ft respectively. The tendency for the three curves to show a uniform increase in resistivity with depth indicate a fairly sound rock in the area east of the borehole. Station 2 was located at the sinkhole with bedrock interpreted at a depth of 15 ft but possibly at a depth of 21 ft. Station 1 indicated bedrock at 6 ft. The profiles for stations 1 and 3 show a sudden change in apparent resistivity trends at depths of



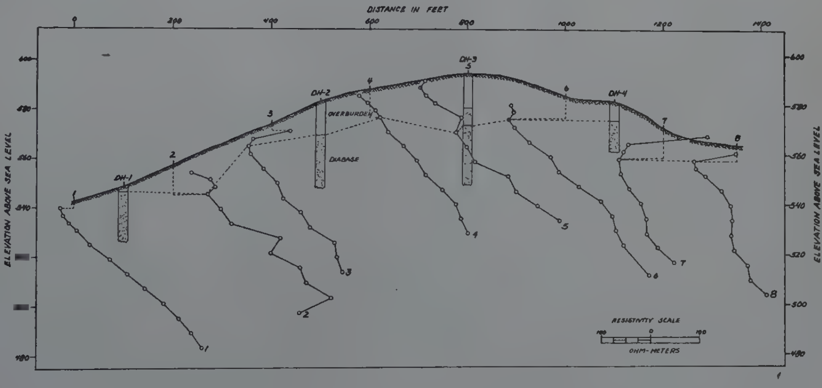


Fig. 4—Cut south of Lawn, Lancaster County, Philadelphia Extension, Pennsylvania Turnpike.

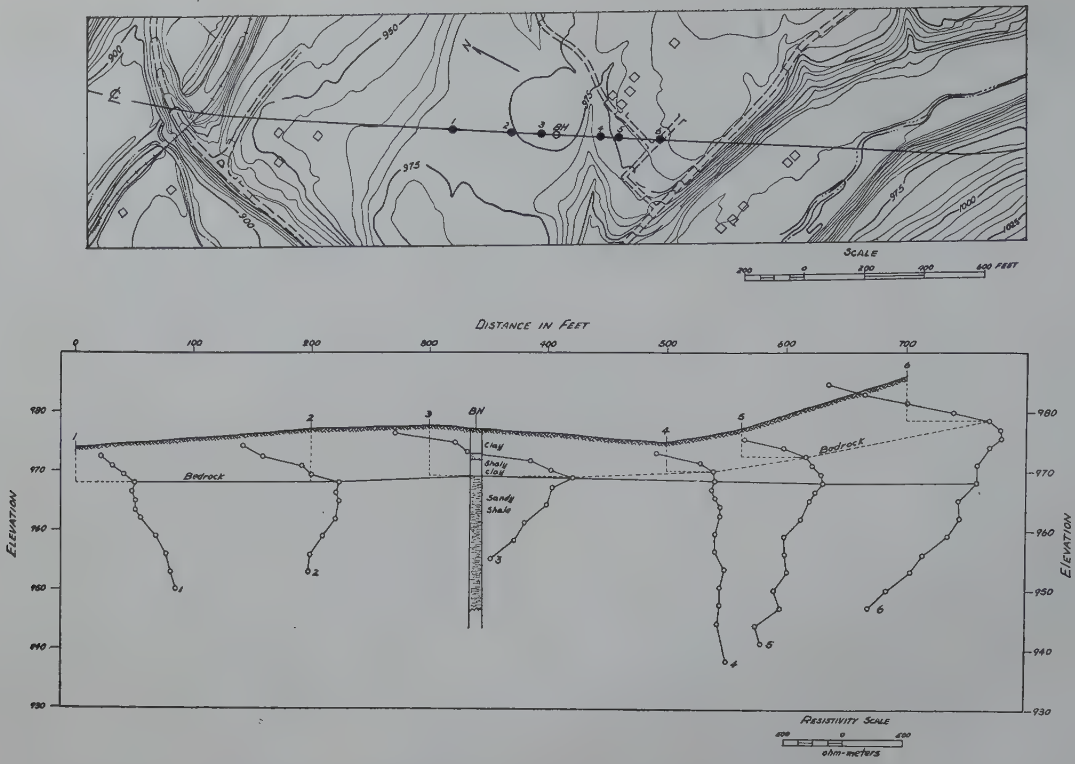


Fig. 5—Cut near Unity, Allegheny County, Western Extension, Pennsylvania Turnpike.

30 and 21 ft respectively. These breaks may reflect cavernous conditions in the limestone at these depths but most likely reflect the sinkhole area in the vicinity of station 2.

Similar analyses have been used in other limestone cuts along the centerline proving beneficial to preliminary investigations prior to actual excavation.

Upper Triassic Shales and Sandstones

In a cut adjacent to Furnace Run in the vicinity of Route 501 and west of Hopeland, Lancaster County, shales and sandstones of the New Oxford formation occur, see Fig. 3.

Two holes were drilled in this cut, recording an overburden of sandy and silty clay with occasional rock fragments and stray diabase boulders. Bedrock is sandstone at depths of 22 and 12 ft respectively.

The apparent and summation apparent resistivities for stations 4 and 6 were computed and correlated against the two drill holes as shown. Breaks on the apparent resistivity curves are not evident; however, breaks on the summation resistivities can be correlated against the drill results. These summation profiles were used principally as a means of interpretation of the 16 different points of observation.

A generalized cross-section indicating depth to bedrock is shown. From stations 1 to 10 the cut has been opened, and correlation between the actual and interpreted depths to bedrock was found to be remarkably in agreement within the measuring interval, here 3 ft. The stations 11 through 16 are in a fill area rather than a cut. Usually in areas of sandstones, the break between overburden and bedrock is sharp.

Table I. Reliability of Geophysical Method

Cut	Length, Ft	Geophysical Report, Ft	Actual Depth to Rock, Ft
A[a]	1500	10 to 13	8 to 9
B	200	No rock above grade	No rock
C[a]	300	9 to 10.5	No rock, 8 to 10 cut
D	700	8 (avg)	No sound rock. Compact shale and rotten sandstone varying in depth from 3 to 12.
E[a]	500	3 to 9	3 to 8
F[a]	550	9 to 17.5	10 to 13 (?)
G	1600	No rock	No rock
H	2900	12 to 18 (15 avg)	7.5 to 22.5 (15 avg)
J	1400	No rock	13 to 15 (?) Probably very little rock in cut.
K[a]	1600	No rock	No rock
L[a]	1525	7.5 to 21.0 (16.5 avg)	Shovel removing rock at average depth of 7 to 8

[a] These cuts investigated by electrical resistivity method without the aid of borehole data.

Upper Triassic Igneous Rocks

The highway centerline just south of Lawn, Lancaster County, crossed a zone of diabase intruded in the form of a sill into Triassic shales and sandstones parallel to the original bedding. All of the strata strike apparently N45E and dip between 40° and 45° northwest. The centerline over this rock roughly parallels the strike, as shown in Fig. 4.

An observation of the eight electrical resistivity profiles in this diabase area show changing resistivity characteristics from profile to profile. This was considered at first as reflecting strong jointing and fracturing. Interpreted depths to bedrock varied from station to station. Later, when the cut was excavated, it was found that depth to sound rock varied considerably and for the most part was much deeper than that recorded by the electrical measurements. The diabase, where deeply weathered, leaves a relic soil with extreme variations in moisture content. Within these relic soils unweathered, isolated remnants and "cupolas" remain as reasonably fresh rock. The drilling encountered these rather than the average depth level of solid rock. The excavation did confirm the resistivity results as to the thickness of the clay overburden and the contact of the diabasic

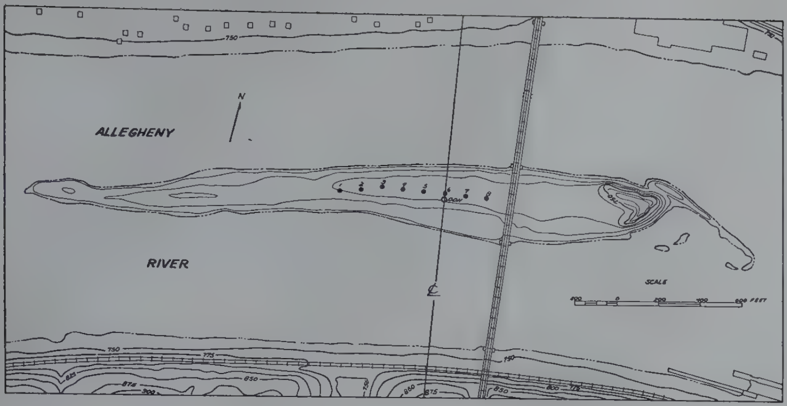
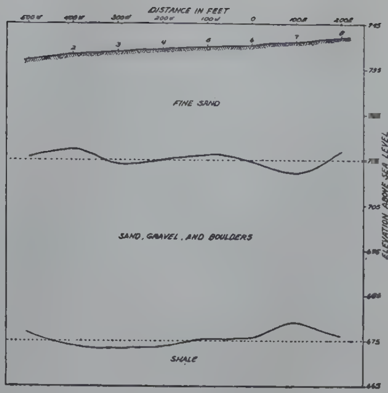
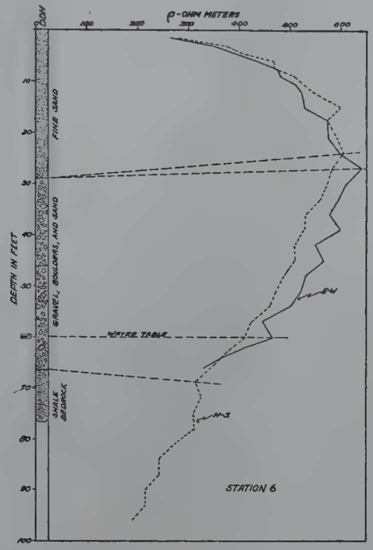


Fig. 6—14 Mile Island Bridge site, Allegheny River.





relic soil. However, the electrical work from a practical point of view was not helpful. It would be impossible to distinguish these fresh remnants in the relic soil by electrical measurements.

Even though the results were negative from a practical point of view, it did lead to a more detailed drilling program and critical review of other diabase areas.

Pennsylvanian Shales—Conemaugh Formation

A small proposed cut involving nearly horizontal strata of sandstones and shales in the Conemaugh formation was investigated near Unity, Allegheny County, see Fig. 5. Drilling of a borehole revealed a bed of sandstone underlain by a layer of shaly clay which was succeeded by a thick column of sound rock starting at a depth of 8 ft.

Six electrical resistivity stations were occupied with the resultant apparent resistivity profiles as shown. The depth to the top of the sandy shale formation was correlated with a down break in the resistivity profile at station 3. This type of break was correlated on the other profiles; however, it is believed that the sandy shale does not constitute bedrock throughout the entire cut. From stations 4 through 6, the depth to bedrock decreases, as witnessed by the correlation of the point shown by the broken line. In this interval, the sand horizon becomes thicker and the shaly clay pinches out, which is a common occurrence in these formations.

This cut has not been excavated to date; however, in areas of flat-lying rocks without the influence of pronounced structural features, it is reasonable to assume that the correlation of major resistivity "breaks" with stratigraphic units shown by core boring data are fairly accurate. A few of the cuts near Irwin have been opened with a remarkable agreement, within the measuring interval, between actual and interpreted depths to bedrock (about 1½ to 3 ft).

Recent Gravels and Pennsylvanian Shales

The centerline of the Western Extension crosses the Allegheny River at 14 Mile Island, as shown in Fig. 6. A high bridge crosses the river above this island, hence, knowledge of the depth of bedrock was essential.

A drill hole was drilled on the centerline with the log showing one horizon of fine sand succeeded in depth by a thick section of gravel, sand, and boulders with shale bedrock at a depth of 66 ft.

Eight resistivity station centers were located with the electrode configuration extending in east-west and north-south directions so that at each station two resistivity profiles were obtained. The profiles at station 6 are shown correlated with the drill log. Very definite breaks on the resistivity profiles correlated precisely with horizons in the drill log. On the basis of these characteristic trends, a vertical geological section was made along the line of resistivity stations showing the distribution of these sands, gravels, boulders, and depths to bedrock.

Reliability of Geophysical Method

Over Upper Triassic shales and sandstones, 11 cuts have been opened. A careful analysis has been made of the actual geological conditions and those reported on the basis of the electrical work. Table I illustrates the reliability of the geophysical work. It demonstrates that most of the interpreted depths were within the limit of error, which was 3 and 6 ft.

Caution is advised in acceptance of the electrical resistivity method as a complete tool by itself. The importance of basic geologic field work and corroborative core-borings is essential in such geophysical programs when applied to highway construction.

The accuracy of the electrical resistivity method in determining overburden thickness in the extensions of the Pennsylvania Turnpike System, as demonstrated in 104 cut sections (some excavated and all checked by core-boring data), is approximately 1.88 ft. This accuracy, in effect, takes the gamble out of bidding on "common" excavation for the contractor and is reflected in a low bid price with consequent saving to the owner.

Development of the South American Coal Industry

by Thomas Fraser

There are large coal reserves in Argentina, Brazil, Chile, Colombia, Peru and Venezuela, and growing productive industries in Brazil, Chile, Peru and Colombia. The paper describes current activities in these coal fields, and improvement programs that are in progress, or are planned. Special emphasis is placed on improvements in mining and preparation practice, and expansion of production that have been accomplished by American equipment and technical assistance.

Up to the present, there has been no large coal-producing industry in any of the South American countries. Since colonial times, the great mineral and agricultural wealth of that continent has led the people to activities other than the development of manufacturing industries based on coal; and climatic conditions are such that there has been very little demand for solid domestic fuel. However, there are extensive and widely distributed reserves of coal in South America that would make the development of a modern coal industry feasible. That factor is of the utmost significance now, when it is believed that the South American countries are entering the pioneer phase of a broad industrial development.

Coal Fields

Great areas of coal-bearing formations of Cretaceous and Tertiary ages occur generally throughout the Andean uplift and extend into the coastal range of the Caribbean region far east of Caracas. Thus the west coast countries have adequate reserves of coal, which include a wide range of types and classes.

In the eastern part of South America, east of the Cordillera, as in North America, the coal measures are of Carboniferous and Permocarboniferous ages. The principal known deposits in this area are in the southern States of Brazil. Permocarboniferous coals ranging in rank from lignitic to anthracitic are found in the States of Rio Grande do Sul, Santa Catarina, Parana and Sao Paulo. In the great interior plateau of Brazil, south of the Amazon River, are extensive areas of Carboniferous strata that may contain coal deposits. There are widely scattered reports of findings of coal in wells and stream beds in northern Piaui and Goias. In the Rio Fresco Valley, explorers have observed outcrops of two beds of anthracite and have brought to Rio de Janeiro samples from these beds. The Fresco is a tributary of the Xingu River that enters the Amazon from the south, but coal could not be shipped by water since the Rio Fresco area is in the upper part of the Xingu Valley, and there is a formidable series of rapids between that area and the lower level of the Amazon River basin. It is doubtful if coal deposits in this general area will be of economic importance for many years to come.

Fig. 1 shows the general location of known coal fields of South America that have been explored sufficiently to indicate the presence of workable reserves.

The coal deposits of the Andean region are similar in occurrence and general characteristics, as well as in age, to the coals of the Rocky Mountain region of North America. Although the coal fields obviously

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must be greatly disturbed by the tremendous Andean upheaval, there are evidences of large blocks of coal persistent over substantial areas. This applies particularly to explored areas in the coal fields of Colombia and Venezuela.

The coal measures of Cretaceous and Tertiary ages in the Andean area have been subjected to great tectonic influences, and there is even a wider variety of types of coal than in our Rocky Mountain section. On the other hand, the coal beds in the eastern part of South America are relatively less disturbed and are now persistent and uniform in thickness; in many locations they are substantially level.

Peculiarly, the similarity between the South American and North American coal areas is manifested further in the distribution of sulphur in the coal. The Andean coals are, for the most part, low in sulphur, presenting nowhere any difficult washing problem with respect to sulphur reduction, although they may vary greatly in ash content. On the other hand, sulphur is an important impurity in the Brazilian coals, and the characteristic occurrence of the sulphur varies greatly from area to area, as is so true of the coal fields of the Appalachian region of North America. For example, in the coking-coal region of the State of Santa Catarina, the sulphur often amounts to as much as 10 or 12 pct in the raw coal, but it can be reduced to 1.50 pct by washing—a remarkable washing performance. On the other hand, the coal deposits in the Rio de Peixe region of northern Parana, varying greatly in sulphur content, may have a large proportion of the sulphur in the organic and finely disseminated pyritic form. These coals do not respond well to the washing operation for sulphur reduction.

Reserves

In the present state of exploration and development of coal in South America, it would be futile to attempt to set up quantitative statistical data on tonnages of coal available in that continent. The coal resources of South America have only recently received serious attention, except in a very few localities, and the extreme difficulty of exploration in a large part of the area in which coal deposits occur makes it unlikely that thorough exploration and appraisal of reserves will be accomplished in the near future. In fact, with the present outlook for practical development of the coal industry and the expansion of the market in South America, such an expensive undertaking would not be justified. In most instances where coal is currently being produced, forward exploration work is sufficient to insure the availability of reserves adequate for present needs.

For these reasons, it is more practicable and probably more useful to treat the subject of reserves on a relative basis by classifying the South American countries and industrial communities with respect to the adequacy of available coal reserves, taking into account not only the information available with respect to the extent of coal deposits, but also the prospects of expansion of the need for the product over the foreseeable future.

Argentina, Brazil, Colombia, Chile, Peru, Ecuador and Venezuela very probably have adequate reserves of coal that can be developed to meet the needs of those countries over a long period of time, visualizing a normally rising curve of industrial activity; and these countries can supply coal to the international trade among the South American countries.

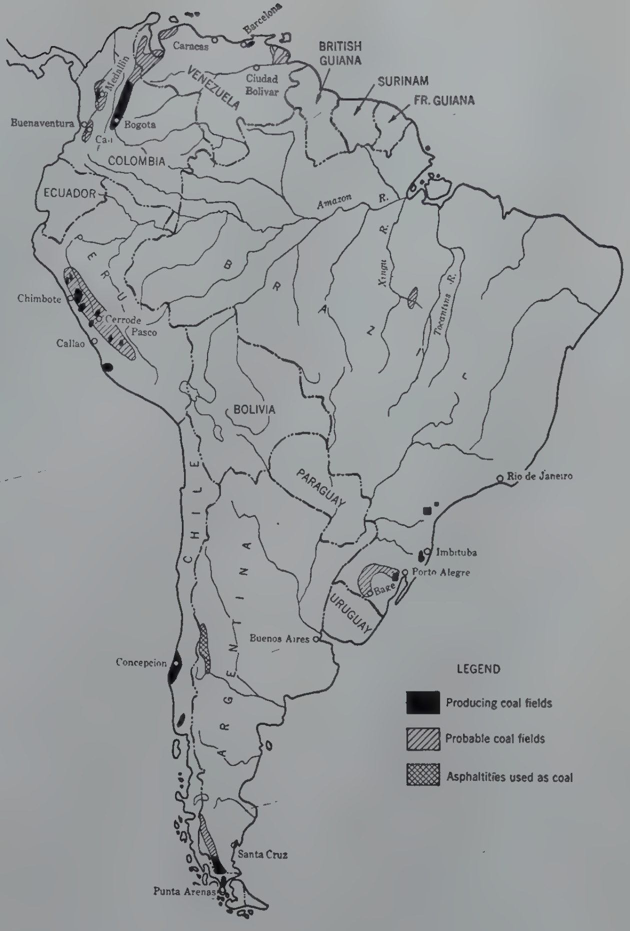


Fig. 1—Principal coal-producing fields of South America.

Of these countries, only Colombia, Peru, and probably Venezuela have adequate reserves of coal in a range of types and kinds that meet the needs of those countries in the development of a diversified industry over the reasonably long-term future.

Argentina, Brazil and Chile, on the other hand, while having adequate quantitative reserves, probably lack high-grade, special-purpose coals that would be needed in a modern industrial state. These countries must continue to import part of their coal needs for such purposes as gas-production and metallurgical operations.

Some occurrences of coal in Bolivia, Paraguay and Uruguay have been reported, but there are not at present known deposits of workable coal in those countries that would justify development of a mining industry to supply their fuel needs.

State of Industrial Development

Some recent data on the rate of production of coal in South America are presented in Table I. While most of the South American countries have ade-

Table I. Rate of Coal Production in 1948

Country	Tons
Argentina	62,000[a]
Brazil	2,013,000
Chile	2,239,000
Colombia	1,000,000[b]
Peru	187,000
Venezuela	21,000

[a] Asphaltites used as fuel.
[b] Estimated.

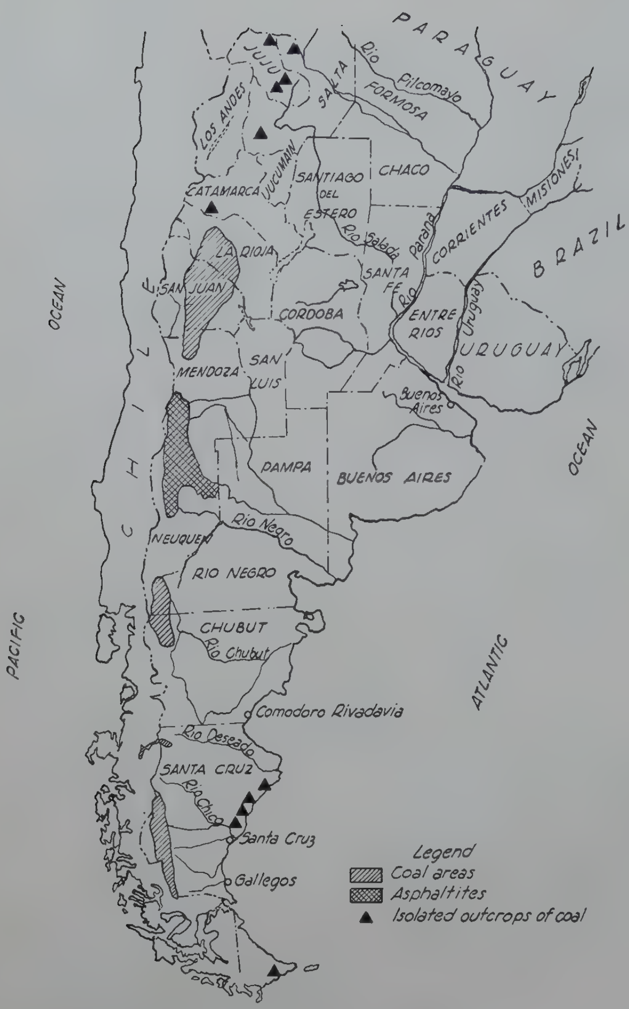


Fig. 2—Coal deposits of Argentina.

quate reserves of coal that would enable them to meet their fuel requirements from national production, only Chile, Colombia, and Venezuela currently are doing so. Chile and Peru have, from time to time, exported some coal to other South American countries, but on balance Peru generally has not produced as much coal as is consumed in that country.

In a few localities, namely the Arauco Bay area of Chile, and in the Sao Jeronimo area of the State of Rio Grande do Sul, Brazil, the coal-producing industry dates back into the 19th century. In these localities there are a few relatively large-scale producing operations with well-organized, systematic methods of mining and some mechanical equipment. Even in those countries, however, the small market for coal heretofore has not fostered any substantial expansion of the coal industry. Generalizing with respect to South America as a whole, the coal industry is in a pioneer stage, just entering upon the era of expansion. It may be considered, therefore, to be a virgin field of industrial activity. There are extensive reserves of virgin coal, and conditions favor the development of modern systematic coal-mining operations, starting without any inherited obsolete plant or practice. However, there are certain economic and social conditions that have a profound influence in practical developments. While coal mining in South America is decidedly in the pioneer phase with respect to physical operations, traditional methods of financing and organizing industrial enterprises are European rather than American. There are many obstacles to the organization of productive work in accordance with American practice to obtain large-scale, low-cost operation.

Most of the South American countries are well-advanced in the direction of liberal labor laws and social programs. While these programs, in most instances, come far short of complete enforcement in industry, they constitute a factor of substantial importance in planning a long-term industrial development.

This advancement of social legislation, far ahead of the actual realization of the complex industrial state to which it is adapted, presents an environment much different from that in which the beginnings of mechanized industry were successfully undertaken in the United States. Young industries will have to carry a relatively heavy burden of welfare expenses and regulatory restrictions during the critical proving phase, but these factors may be compensated by government supports and technical aids that were unheard of during the pioneer period of American industrial development.

With respect to governmental attitude toward the development of the coal industry in South America, almost every condition can be found ranging from complete nationalization of the industry on one end to entirely independent, free-enterprise operation on the other. In Colombia the annual production of coal, amounting to approximately one million tons, comes from at least 200 competing small producers; and it is still possible to start a coal-mining enterprise in that country with very little financial backing.

There is another factor that has a very great practical bearing on the development of South American coal industry; unlike the step-by-step pioneer development of the coal industry in the United States, expansion in South America is beginning at a time when prefabricated, complex, modern physical plants and production systems are being transplanted full-grown directly to the frontier.

Current promotional planning in South America is very largely directed to this type of development. Indeed, this concept is even more prominent in broad planning of general industrial development and is currently popular throughout the world. It is to be expected, therefore, that the South American industry will adopt the popular philosophy. In undertaking to apply American industrial knowledge, there is the problem of developing the most effective introduction of American technique under strange and varying social and economic conditions. On the one extreme, there is the traditional American pioneer method of modest beginning followed by gradual expansion, articulated with concurrent development of other industries as the general capabilities and needs of the community develop. On the other extreme, there is the forcing technique by which an attempt is made to advance a backward community in one leap to the ultimate in modern mechanical industrialization, using large capital sums supplied by government or government-supported agencies. The pioneer method would appear to be the more conservative and more certain of ultimate success. On the other hand, the forcing technique appears to offer the possibility of saving precious time in the development program—at a price. Many experiments are currently underway that should show, over a period of years, how real is this time saving and at how great a cost. Both

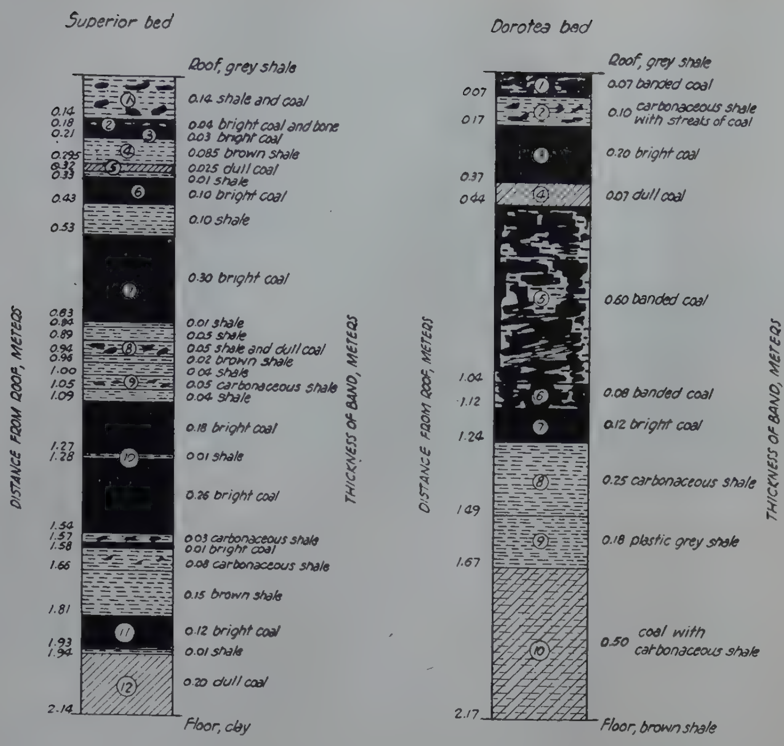


Fig. 3—Typical sections of coal beds in Rio Turbio field, Argentina.

types of development programs are in operation in the South American coal industry today, and there would be no better field in which to test the relative merits of these two systems by actual performance.

In South America, a modern industry must be built on a base somewhat different from ours—a base in which the principal elements are a new world wealth of natural resources; a new world frontier physical environment, and a people with an advanced "old-world" philosophy of life.

Transportation Facilities

In any coal-producing country, activity starts first where transportation facilities are available. Coal deposits in remote, inaccessible areas, no matter how extensive and no matter how high in quality, cannot be considered of any immediate economic value. This is especially important in studying the coal reserves of South America; the extensive coal-bearing measures are in the Andean countries, and there the terrain is so difficult for railway and highway construction that a relatively few miles of mountain terrain may constitute a definite obstacle to the development of a rich coal field while there is a relatively small potential market for the product.

Hence, the location of railways, which in colonial times were built to serve agricultural areas, and proximity of tidewater ports have been the determining factors in local development of small coal-mining industries. In South America this interdependence is intensified since the railways themselves furnish the most important market for coal.

Table II. Coal and Asphaltite Consumption in 1948, in Metric Tons

	Coal	Asphaltite
Railway fuel	673,000	8,000
Public offices	52,000	18,000
Utilities	614,000	7,000
Meat packing	67,000	
Industrial	45,000	14,000
Miscellaneous	126,000	5,000
Total	1,577,000	52,000

In several South American countries, more than half the total national consumption of coal is in the category of railway locomotive fuel, despite the tremendous quantities of wood used for firing locomotives. This predominance of the railway in the coal market perhaps will increase in the future as the expected increase in traffic and growing scarcity of timber force conversion of wood-burning locomotives to other types of fuel. Because of this dependence of coal-mine development upon the availability of local transportation, there are many instances of productive activity in coal deposits of minor extent and substandard quality explainable entirely on the basis of accessibility.

There are only five active coal-producing areas that are accessible to tidewater ports and in a position to export coal or to move coal from one point to another by sea. These five coal-producing areas are: 1—The central coal field of Chile adjacent to Arauco Bay; 2—the Santa River Valley anthracite region connected to the Pacific coast port of Chimbote by railway; 3—the Sao Jeronimo coal field of Rio Grande do Sul, Brazil, which ships coal to tidewater by the Jacui River; 4—the Santa Catarina coal field of southern Brazil, connected by railway to the Atlantic coast ports of Imbituba and Laguna, and 5—the Naricual coal field of Venezuela, connected by rail and by highway to the Caribbean seaport of Guanta. Substantial tonnages of coal have been dispatched through that port in the past, but the field is not operating at present. The Cauca Valley coal field of southwestern Colombia is served by the railway of the Pacific, which connects with the Pacific coast port of Buenaventura. However, no coal has actually moved out through that port for export. The

Table III. Sources of Energy in 1945, Pct

Petroleum and its products	28.1
Coal and coke	6.0
Wood and charcoal	34.4
Grain and farm waste	26.2
Gas	4.8
Hydroelectric power	0.4

Fig. 4—Miner working at face, Galeria No. 1, Rio Turbio Coal field of Argentina.

Photograph courtesy of U. S. Bureau of Mines.

Magallanes coal fields of southern Chile have a ready outlet through Punta Arenas, but no coal has been exported since early 1945, when the principal mine in that area was shut down. Exports before that time were to Argentina. Except for these fields, all capable of relatively small production viewed from our standpoint, the great reserves of coal in South America are in the interior and are currently being worked only in relatively small local areas to meet internal market demands. In most localities, the interdependence of the railways and the coal industry is very sensitive. Some railways are chronically short of fuel, and everywhere it is true that even a relatively small increase in railway traffic immediately creates a stringent bottleneck in the form of locomotive-fuel shortage. Hence, although coal is not generally recognized to be a strategic mineral, it almost immediately moves into the position of number one transportation problem in any general attempt to increase the rate of strategic mineral production.

Notwithstanding the relatively unimportant position of South American coal production in the world picture, the industry is growing in importance in almost every country; there have been some outstanding developments during the past 10 years, and many new projects are currently under development.

Argentina

There is no other South American country in which fuel shortage is so serious as in Argentina. The Argentine government is making determined efforts to develop a coal industry in the Rio Turbio coal field in the southern part of Patagonia.

Fuel Needs: Current needs of Argentina for coal and other solid fuels (except wood) are indicated by the consumption in 1948, Table II.

Currently, coal does not furnish a very large proportion of the total energy consumed in Argentina; but, if there is to be a marked industrial expansion in this country in the future, the energy load must be shifted in large measure to coal, as it is not to be expected that the supply of wood and agricultural products available for fuel is capable of any substantial expansion. Statistical data available up to and including 1945 indicate that, in round numbers, the total national consumption of energy increased from the equivalent of 9,657,000 tons of petroleum in 1939 to the equivalent of 12,500,000 tons of petroleum in 1945. Sources of energy as of 1945 are shown in Table III.

Practically all the coal consumed in Argentina is imported. Currently, this imported coal is furnished by Great Britain under the provisions of a commercial treaty recently completed between Argentina and Great Britain. The two central power stations of the Cadi Co. in Buenos Aires are currently receiving British coal at the rate of about 2000 tons daily. Although coal does not supply a very large proportion of the total energy used in Argentina, the importation of coal requires the expenditure of a substantial proportion of the foreign exchange available to Argentina. It is of the utmost economic importance to replace this imported coal, transported over a very great distance, by national production so that these large sums of foreign exchange may be diverted to the purchase of industrial machinery and miscellaneous manufactured goods for Argentina's expanding industries.

Coal Production: For many years the total production of solid mineral fuels in Argentina has been around 60,000 to 70,000 metric tons annually and practically all of this consists of asphaltites derived from mining operations in Mendoza and Neuquen.

There are also some small deposits of lignites and bituminous coals in these States and scattered in several parts of Argentina; but the only substantial reserve, so far explored, is the Rio Turbio field in the southwestern part of the territory of Santa Cruz. Fig. 2 shows the location of the known occurrences of coal deposits in Argentina.

Some years ago the Rio Turbio field was opened by agencies of the Argentine Navy, and a producing mine was placed in operation near the main camp established by the Navy on the Estancia Dorotea near the Chilean boundary. At present, the expansion of operations there is in the hands of a separate Government agency called Direccion General de Combustibles Solidos Minerales, and an ambitious five-year plan has been projected to establish a modern coal-mining industry in that area; to improve the port of Santa Cruz on the Atlantic coast of Argentine Patagonia to the north-

Table IV. Typical Analyses of Face Samples

Analyses	Dorotea Bed, Galeria 34	Superior Bed, Galeria 25
Moisture, pct	7.4	7.0
Volatile matter, pct	30.6	34.0
Fixed carbon, pct	44.0	46.0
Ash, pct	18.0	13.0
Sulphur, pct	1.67	
Calorific value, Btu	10,900	11,100

Table V. Brazilian Coal Production[a] and Imports,[b] 1940 to 1946, Metric Tons

Year	Brazilian Production	Imports	Total	Coke Imports
1940	1,048,533	1,185,904	2,234,437	23,338
1941	1,109,835	1,033,323	2,143,158	24,623
1942	1,354,089	592,761	1,946,850	23,277
1943	1,537,432	712,562	2,249,994	52,239
1944	1,414,607	709,957	2,124,564	12,208
1945	1,489,666	956,980	2,446,646	17,920
1946	1,273,708	1,002,620	2,276,328	11,506
1947	1,421,216	1,531,111	2,952,327	41,621
1948	1,423,399	1,060,150	2,483,549	22,817

[a] Cleaned coal available to consumers.
[b] Comparative heating values: 5 tons imported = 7 tons domestic.

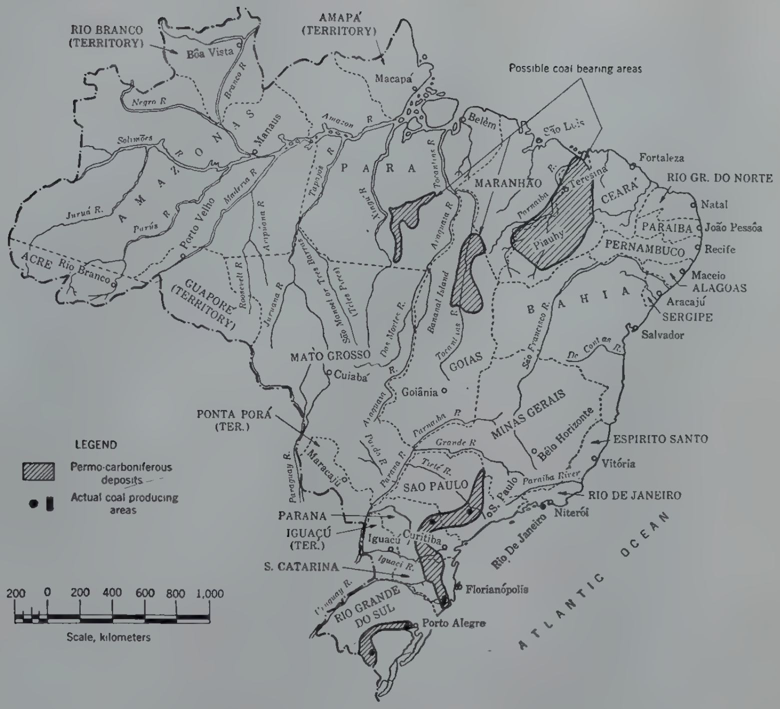


Fig. 5—Coal fields of Brazil and known producing areas in the southern states.

east of the coal field; and to construct a railway to connect the coal field with that port. There is now an all-weather highway connecting the mining area with the port of Rio Gallegos on the Patagonian coast to the south of Santa Cruz; some coal has been shipped from this port to Buenos Aires and used in the Cadi power plants.

Rio Turbio Field: The Rio Turbio coal field is an area approximately 30 to 40 km long in a generally north to south direction and 15 km wide at the south and about 8 km wide at the north. It extends from near the Chilean border north of Natales on the south to the Estancial Cancha Carreras on the north. There are at least three coal beds of good working thickness, varying from 1½ to 3 m; and that part of the field that has been adequately explored by trenches, development workings, and drill holes is estimated to contain approximately 500,000,000 tons of recoverable coal.

These coal deposits are considered by geologists of the Direccion to be of early Tertiary age in the formation locally called the Patagonian. Presumably these coals are substantially of the same age as the deposits extending throughout the Andean region; these coal-bearing measures are in some of the countries considered to be late Cretaceous and in other countries they are designated as early Tertiary. The Rio Turbio coal beds outcrop along the western margin of the basin extending over the international boundary line into Chile in the south part of the area. Drill hole records show that the beds terminate underground along the eastern margin, indicating that this margin is the natural boundary of the basin in which the coals were originally laid down. However, exploration work has not proceeded far enough to prove definitely this hypothesis or to delineate the eastern extent of the basin precisely. It is held to be probable that there are other deep coal basins to the east, since the Patagonian extends to the Atlantic coast in that area and attains great thickness.

Typical sections of the two most valuable coal beds are shown in Fig. 3, and some examples of analyses of face samples of the coal are presented in Table IV.

A series of tests of the preparation characteristics of these coals carried out in the laboratories of the Direccion in Buenos Aires indicates that the Dorotea coal is somewhat more responsive to preparatory treatment than the coal from the Superior bed, but both coals may be readily prepared to serve as general purpose fuel.

It is the comprehensive plan of the Direccion, over the next five years, to develop several large, modernly equipped producing operations in the southern part of the Rio Turbio coal field and to build railway yards, shops, power plants, and other surface facilities around the terminal of the projected railway, which will be located in the Rio Turbio valley on the Estancia Dorotea.

There are now three underground operations in that general area—Galeria 1 in the Superior bed, Galeria 25 in the Superior bed, and Galeria 34 in the Dorotea bed. Galeria 1 is the original Rio Turbio mine opened by the navy. It is now on a production basis, worked entirely by hand, winning the coal at the face of chambers turned off the main Galeria and advancing to the rise. The coal picked from the breast by hand is delivered by gravity chutes to cars of

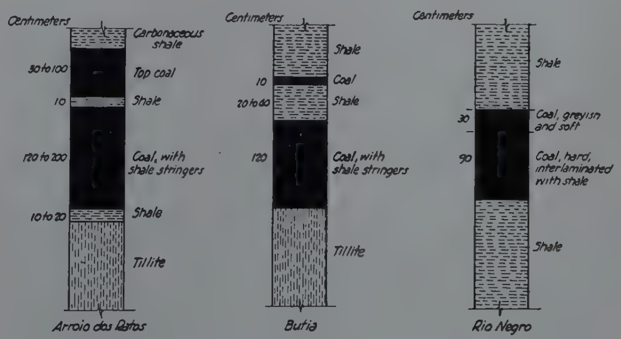


Fig. 6—Typical sections of coal beds in Rio Grande do Sul.

All dimensions in centimeters

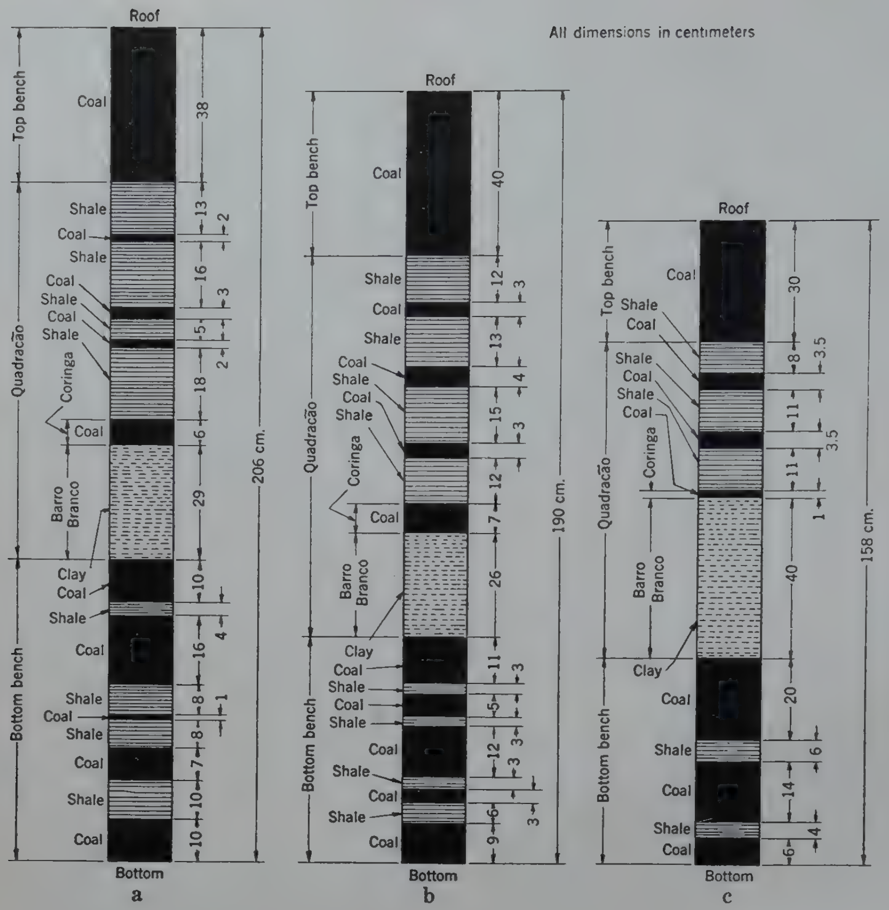


Fig. 7—Typical sections of Barro Branco bed in Santa Catarina, Brazil.

about 1000 kg capacity on the haulage road. Fig. 4 shows a miner working at the face of one of the chambers of the Galeria No. 1 mine.

Galerias 25 and 34 are in the development phase only and are designed eventually to open up important areas in these two coal beds near the proposed railway terminal.

During construction of the railroad, the general improvement program and development of mine workings will be carried on, and the coal produced during that period will be transported to the port of Rio Gallegos by steam lorries. The railway location survey has been completed, and it is currently reported that railway construction has been started.

Brazil

The coal-producing industry of Brazil is in the three southernmost States of that country—Rio Grande do Sul, Santa Catarina and Parana. There has been some small production in the State of Sao Paulo, but the known reserves there are so small that this State is not generally considered to be a potential coal producer.

As in Argentina, the largest load is carried by wood, and coal furnishes only a relatively small proportion of the total energy consumed. In fact, this condition is common throughout South America, with very few exceptions. In any sustained expansion of industrial activity in Brazil on a substantial scale, the increase in total energy would have to be supplied very largely by coal. Firewood production is generally considered to be at about the maximum that it can attain; and in some localities where the railways have depended for many years upon firewood for locomotive fuel, the supply is becoming scarce and costly. There are tremendous waterpower potentials in Brazil, but for a long time the nation's industries and its railway transportation system will be so widely dispersed that the capital investment required to serve these energy users with central station electrical energy would be prohibitive. Hence, a dependable source of coal supply of gradually increasing volume is of great importance to the future development of the economy of Brazil.

The coal needs of the nation have for a long time been supplied from domestic and imported sources, roughly in equal proportions, although this ratio varies from time to time as available foreign sources of coal have varied. Table V summarizes the available statistical data on the consumption of coal in recent years, broken between foreign and national sources of supply.

Reserves: The location of known coal reserves of Brazil is shown in Fig. 5. In some of the established producing areas, parts of the field have been explored by means of drill holes and exploratory openings, and these data have formed the basis of estimates of known reserves ranging variously from 380 million metric tons to 650 million metric tons. The data given in Table VI broken down by States are considered to be conservative.[1]

The data for the State of Santa Catarina refer only to the coal known to exist in the Barro Branco bed; no estimates are included for the Rio Bonito bed or for possible extensions of the Barro Branco bed beyond the areas explored by companies now producing. Estimates for Rio Grande do Sul likewise contain only the relatively well-known areas, and it is very probable that the conservative figure given here would be greatly expanded if exploration work were continued in the Rio Negro field near Baje and in the Condeota valley.

Developments in Rio Grande do Sul: Brazil has one of the oldest coal-mining industries in South America, dating back to the opening of a small mine in Sao Jeronimo County in the State of Rio Grande do Sul in 1853; coal-mining activity has been carried on intermittently in that area until the present. At the beginning of World War II, there were two well-organized, sound mining operations in that field, one at the original mine location of Arroio dos Ratos and the other at Butia. Both operations are under the administration of the Consorcio Administrador das Empresas de Mineracao (CADEM) organized 1916.

These are the largest coal-mining operations in Brazil. The two mines are relatively near to one another in the eastern extremity of the crescent-shaped field shown on Fig. 5 just west of the city of Porto Alegre. In that locality the coal beds are not continuous, but the deposits occur in a series of isolated basins that attain a thickness of 5 to 9 ft in the central part, but thin out toward the edges. Fig. 6 shows typical bed sections at these mining properties and in the Rio Negro field in the south part of the State.

The deposits are opened by shafts and slopes, and all the coal is produced in underground mining operations.

At Butia, the room-and-pillar mining system is used, with some modifications. The use of mechanical mining equipment is limited to pneumatic puncher-type cutting machines and pick hammers. The cutting machines are generally used only in driving narrow work, while the pick hammers are employed in rooms as well as development headings. Endless rope haulage systems are used for underground transportation on the main haulage roads and the principal secondary roads. Loaded and empty cars are hand-trammed between collecting points and the face.

A modified longwall panel mining system is employed at the Arroio dos Ratos mine. The mechanical equipment underground is similar to that at Butia, except that electric trolley locomotives are used on the main haulage roads. The mine workings at both properties are ventilated by exhaust fans of the centrifugal type.

On the surface both these mining properties have modern American coal-preparation plants designed and built by the McNally-Pittsburg Manufacturing Corp. Both plants are combination picking table and Baum-jig washery plants adapted to prepare railway fuel and also to make a secondary coal which is delivered to the utilities company in Porto Alegre for use as power-plant fuel on chain-grate stokers and in pulverized-fuel burners.

Power for the productive operations and transportation facilities is supplied by coal-burning thermoelectric generating plants at each of the mines. The largest of the two plants, at Arroio dos Ratos, furnishes power to the municipality of Sao Jeronimo and to the port of Charqueadas.

Table VI. Estimate of Coal Reserves of Brazil, Metric Tons

Rio Grande do Sul	60,000,000
Santa Catarina	400,000,000
Parana	20,000,000
Sao Paulo	500,000
Total	480,500,000

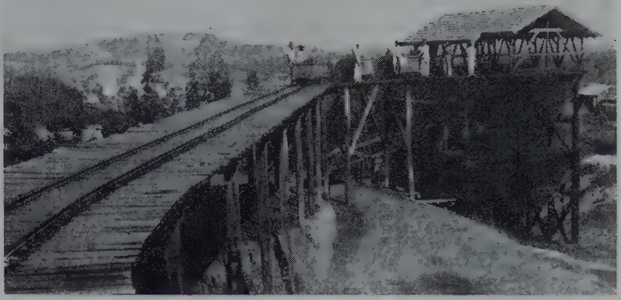


Fig. 8—Hand preparation plant at Bainha mine.

Photograph courtesy of U. S. Bureau of Mines.

At each property, there are general machine shops, blacksmith shops, foundry, carpenter shops, electrical repair shops, and repair shops for rolling stock of the mine-transportation system. At Butia repair work also is done on railway rolling stock.

Black powder, the explosive normally used in the mining operations, is manufactured in company-owned factories at each of the two mines. Normal consumption is about 2640 lb of powder per day at each mine. Each of the two plants has a productive capacity of about 6600 lb of black powder in 8 hr.

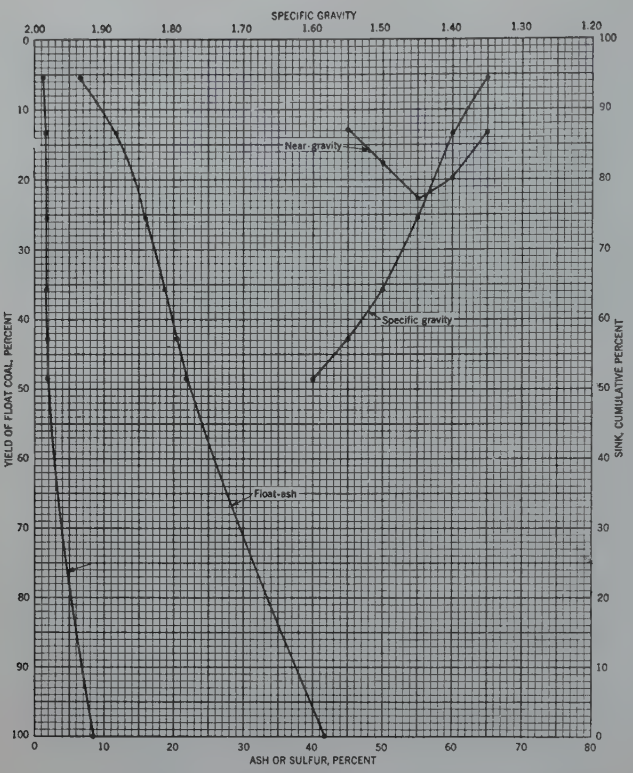
The company also operates a railway to the port of Charqueadas on the Jacui River and has loading facilities there by which coal is loaded into barges for transport to Porto Alegre, from there to be shipped by sea to other consuming areas of Brazil. Surface facilities of the Butia property include a 3600-m aerial tramway across the Jacui River at Condi. This tramway is used to deliver railway fuel to the State railways of Rio Grande do Sul.

This company has an advanced program of social benefits for the working people. There are adequate hospital facilities, schools and housing accommodations. Under the national labor code, the mine operates a 5-day week with four 6-hr shifts each working day.

For many years these two mines furnished the bulk of the national coal production. They reached a peak production record in 1943, when 1,346,269 metric tons was shipped. Since that time there has been a substantial decrease in the production rate, partly because the working areas have advanced into parts of the beds where mining conditions are less favorable, and partly because of the shortened work shifts coupled with a shortage of permanent labor.

To compensate for the threatened shortage of railway and industrial fuel in the State and for the rising costs of fuel, the Government of the State of Rio Grande do Sul in 1947 established a State agency called Departamento Autonomo de Carvao Mineral, under the direction of Jose Borges de Leao, to promote the production of coal on state-owned mining properties. This State agency is organizing a comprehensive program to develop the mining property of Leao in the same general area as the properties of CADEM and is also exploring coal deposits in other parts of the State.

Developments in Santa Catarina: The coal deposits of Santa Catarina, unlike those in Rio Grande do Sul, persist over large areas. The Barro Branco bed is mined extensively in the Lauro Muller, Urussanga, Criciuma, and Sideropolis areas. The Rio Bonito bed below the Barro Branco has not been



Specific gravity fractions	Elementary data				Computed cumulative data						Near-gravity ±0.05 percent
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent			
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur	
Float - 1.35		5.5	6.6	1.39	5.5	6.6	1.39	100.0	41.5	8.40	13.2
1.35 - 1.40		7.7	15.5	1.78	13.2	11.8	1.61	94.5	43.5	8.81	19.9
1.40 - 1.45		12.2	20.6	1.70	25.4	16.0	1.65	86.8	46.0	9.43	22.6
1.45 - 1.50		10.4	25.2	1.96	35.8	18.7	1.74	74.6	50.2	10.70	17.4
1.50 - 1.55		7.0	28.6	2.67	42.8	20.3	1.89	64.2	54.3	12.11	12.7
1.55 - 1.60		5.7	33.0	2.42	48.5	21.8	1.96	57.2	57.4	13.27	
Sink - 1.60		51.5	60.1	14.47	100.0	41.5	8.40	51.5	60.1	14.47	

Fig. 9—Washing characteristics of coal from Barro Branco bed. Size, 2 in. x 48-mesh.

explored, but it is generally thought also to be persistent and substantially level.

The Barro Branco coal bed, in which nearly all the producing mines are working, is a high-rank coking coal of medium-volatile content, but it is so intimately intermixed with shale and bony bands that the preparation of a usable coal is very difficult. Typical sections of this coal bed at operating mines are shown in Fig. 7.

Coal has been mined in this field for many years, but until the establishment of the national steel industry, the production was relatively small compared to Rio Grande do Sul, which dominated the national coal-producing industry. During the period of development and construction of the Government-sponsored national steel industry, a concerted effort was made to expand the production of coal in Santa Catarina, which is the only national source of metallurgical coal. The Brazilian National Government took many practical steps to support the coal industry in that area; and the United States Government, working through the Board of Economic Warfare, succeeded by the Foreign Economic Administration, and later the Bureau of Mines, cooperated actively with the National Government in these efforts, and several technicians furnished by the United States were resident in the coal field during that period. Stimulated by these efforts and by the expanding coal market furnished by the national steel industry, the production of coal in Santa Catarina increased from 268,213 metric tons in 1940 to 980,000 metric tons in 1946.

In 1940 there were a number of relatively small drift mines in the Criciuma area and a few somewhat larger operations, some of which were opened by shafts and slopes, in the Lauro Muller and Urussanga regions.

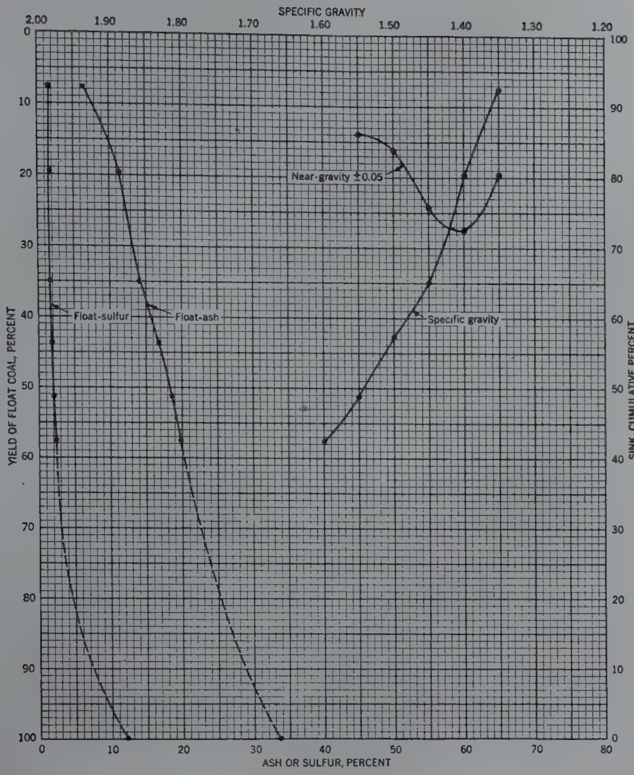
Throughout the area, the coal is won by hand at the face and loaded into small cars of 600 to 900-kg capacity and trammed by hand to the outside, except in the two or three largest operations. Until very recently, there were no central commercial power-generating stations from which the mines could purchase power, and only the largest operations had mine power plants. Because of almost complete lack of mining machinery, machine shops are confined to a few of the largest central units, and repair or maintenance shops at the mine proper are limited to small blacksmith shops for repair of hand tools. At most of the operations the only surface structures are open sheds of timber construction to house the screening and hand-picking operations. However, there are mine preparation plants at the Lauro Muller property of the Companhia Nacional de Mineracao de Carvao do Barro Branco at Lauro Muller, and of the Companhia Carbonifera de Urussanga at Urussanga. The Lauro Muller plant is equipped with piston jigs and coal-washing tables, and the Urussanga plant (of German design) is equipped with Schuchterman and Kramer-Baum jigs with feldspar in the boxes used for cleaning the fines. Both these plants are of small capacity. In addition, the Companhia Brasileira Carbonifera de Ararangua and the Sociedade Carbonifera Prospera each operates one Deister table for cleaning fines, and several hand jig plants are scattered through the field to handle fines at the rate of 0.5 to 1.0 tons per hr.

In the hand-picking plants, the coal is first screened by hand, then picked laboriously by hand on stationary tables and discharged to pit cars or bins from which the coarse coal is hauled, usually by trucks, to railway sidings from which it is again transferred by hand to railway cars. In the hand-preparation plants the coal is transferred from operating step to step by hand.

The total production of the field is gathered usually by trucks to sidings on the Dona Teresa Cristina Railway, which has branches to the three principal areas of the coal field, Lauro Muller, Urussanga, and Criciuma, and is transferred by this railway to the twin sea ports of Imbituba and Laguna from where it goes to market by sea.

National Steel Co.: This was substantially the physical condition of the Santa Catarina coal industry at the time of inception of the National Steel Industry development in 1940.

As a part of that program, the National Government empowered the National Steel Co. to purchase all the coal produced in the Santa Catarina field or such part of it as might be needed, and it established by decree a uniform price schedule that would insure profitable operation to the industry and encourage expansion of production; it empowered the National Steel Co. itself to acquire mining property and develop coal producing enterprises, and it authorized the Departamento Nacional da Producao Mineral to carry on extensive technical investigations in the Santa Catarina coal field, to establish working regulations and direct the use of employment in the area, and to construct water works and an experiment station in the municipality of Criciuma. The United



Specific gravity fractions	Elementary data				Computed cumulative data						Near-gravity ±0.05 percent
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent			
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur	
Float- 1.35		7.6	6.3	1.53	7.6	6.3	1.53	100.0	33.6	12.07	19.5
1.35 - 1.40		11.9	14.8	2.08	19.5	11.5	1.87	92.4	35.8	12.93	27.4
1.40 - 1.45		15.5	19.9	1.86	35.0	14.2	1.86	80.5	38.9	14.54	24.2
1.45 - 1.50		8.7	23.3	2.21	43.7	16.9	1.93	65.0	43.4	17.56	16.2
1.50 - 1.55		7.5	27.0	3.31	51.2	18.4	2.13	56.3	46.6	19.94	13.9
1.55 - 1.60		6.4	32.2	2.49	57.6	19.8	2.18	48.8	49.5	22.49	
1.60 - Sink		42.4	62.2	25.51	100.0	33.6	12.07	42.4	62.2	25.51	

Fig. 10—Washing characteristics of coal from top bench of Barro Branco bed. Size, 2 in. x 48-mesh.

States Government assigned mining engineers and other technicians to cooperate with the Departamento Nacional in this program to stimulate the production of metallurgical coal for the steel industry.

As a result of these measures, new mines were opened and production was greatly increased, especially in the Criciuma area. Power plants, mostly of portable diesel-driven type with compressors, were installed at some of the mines and pick hammers were introduced in the underground work to speed up the production of coal and the extension of development work. Improvements in details of coal-handling operations were brought about in many instances to increase the productivity per man, especially among surface workers. An experimental hand-picking plant was erected by the Departamento at the Bainha mine, and substantial improvements were introduced to eliminate hand methods of transporting from unit to unit. This plant is shown in Fig. 8.

After extensive exploration work in the vicinity of the town of Beluno, the National Steel Co. acquired property there which contains substantial areas of Barro Branco coal adapted to open-cut mining and developed there the Sideropolis mine, which now produces approximately 10 pct of the total output of the Santa Catarina field. A large proportion of this tonnage is produced by strip-mining methods, using mechanical equipment obtained in the United States. Some underground operations are also being developed, and there is a surface preparation plant with large capacity storage bins and mechanical aids to the work that bring about a substantial increase in production per man as compared with the more primitive mine preparation plants described above.

Capivari Washery: Early in the development program, the National Steel Co. sampled the run-of-mine Barro Branco coal from the several coal-producing districts and sent a test lot of some 50 tons to the United States for washing tests and coking tests.

Based on test data obtained in the United States, the National Steel Co. erected a central coal-preparation plant in the railroad yards near the city of Tubarao to prepare the Santa Catarina coal to make metallurgical fuel. This plant site is near the junction point where the branch lines of the Dona Cristina Railway join to connect the coal-producing area with the Atlantic coast ports of Imbituba and Laguna. Here the steel company erected a 400-ton-per-hr washing plant of American design, furnished by the McNally-Pittsburgh Manufacturing Corp.

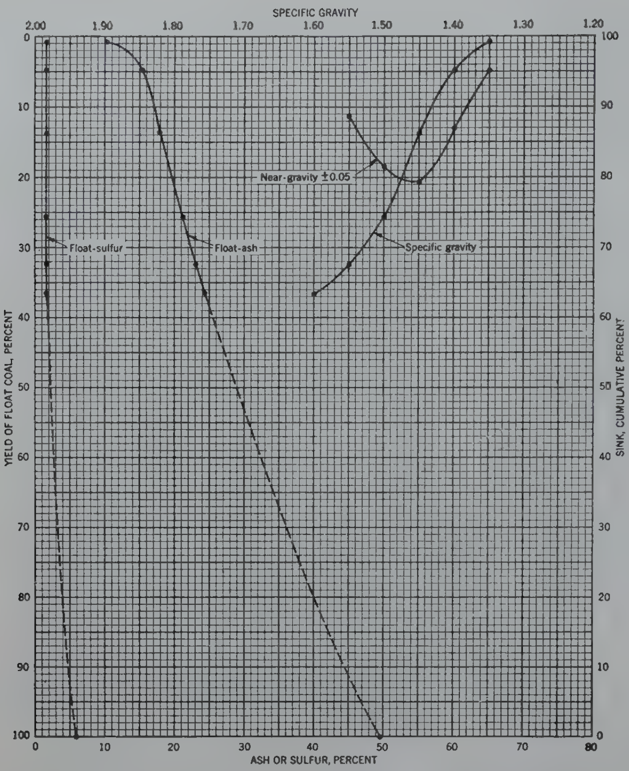
The Capivari washery is a combination Baum jig and Rheolaveur plant housed in a modern reinforced-concrete structure.

There is also a series of raw-coal storage bins in the receiving yards to receive and store raw coal obtained by purchase from the various producers in the Santa Catarina field. Besides serving as storage bins to furnish a continuous flow of raw material to the preparation plant, these bins must also serve as separate storage bins for the different producers. There is also a central power-generating station at Capivari to produce power for use in the preparation plant and for the coal producers in the Santa Catarina field. This plant is adapted to use secondary coal from the washing plant.

The Barro Bronco coal, though a very good coking quality, is extremely difficult to wash, and the yield of metallurgical coal is small. Therefore, it is necessary to make a three-product separation and to dispose of a large portion of the product as a general-purpose fuel coal of relatively higher ash content and lower heat value than the first-grade coal, which goes to the coke ovens of the steel plant at Volta Redonda.

A typical sample of the Barro Branco coal from one of the mines in the Criciuma area shows the preparation characteristics. The data sheet is reproduced in Fig. 9 and indicates the difficulty of preparing this coal for metallurgical use. Currently the Capivari washery is operating to produce a metallurgical-grade coal of about 22 pct ash content and is obtaining a yield of this grade of around 30 pct. Then there is a product amounting to 30 to 40 pct of the raw coal that is used in the power plant or shipped to market for industrial purposes. At the Volta Redonda plant, the Santa Catarina coal is blended usually with around 50 pct of high-volatile imported coal, drawn from eastern United States or Great Britain. Although this blend yields a coal of relatively high ash content, it is nevertheless well adapted for use in the blast furnace plant at Volta Redonda because the iron ore for this plant, obtained from the Minas Geraes iron range, is very high in iron content, and it is desirable to have a relatively large volume of coke ash to supply a slag volume adequate to slag off the sulphur, which runs about 1.5 pct in the coal.

Under the cooperative agreement between the U. S. Bureau of Mines and the Departamento Nacional de



Specific gravity fractions	Elementary data				Computed cumulative data						Near-gravity ±0.05 percent
	Weight grams	Percent			Float, cumulative percent			Sink, cumulative percent			
		Weight	Ash	Sulfur	Weight	Ash	Sulfur	Weight	Ash	Sulfur	
Float- 1.35	125.3	0.8	10.1	1.56	0.8	10.1	1.56	100.0	49.6	5.96	4.9
1.35 - 1.40	680.0	4.1	16.0	1.49	4.9	15.1	1.50	99.2	49.9	6.00	13.0
1.40 - 1.45	1470.0	8.9	19.3	1.64	13.8	17.8	1.59	95.1	51.4	6.19	20.5
1.45 - 1.50	1910.0	11.6	25.1	1.51	25.4	21.1	1.55	86.2	54.2	6.66	18.4
1.50 - 1.55	1130.0	6.8	30.0	1.65	32.2	23.0	1.57	74.6	59.2	7.47	11.2
1.55 - 1.60	830.0	4.4	34.0	2.55	36.6	24.3	1.69	67.8	62.2	8.05	
1.60 - Sink	10470.0	63.4	64.1	8.43	100.0	49.6	5.96	63.4	64.1	8.43	
	16515.3										

Fig. 11—Washing characteristics of coal from bottom bench of Barro Branco bed. Size, 2 in. x 48-mesh.

Producao Mineral, extensive tests of the washing characteristics of the Barro Branco coal have been extended throughout the field. These data have shown that the upper bench of that bed (above the middle bench called the Barro Branco), is relatively cleaner and much easier to wash to metallurgical grade than the bottom bench. It would probably facilitate operations at the Capivari washery if the two benches of the Barro Branco bed were mined separately, the top coal only to go to metallurgical use. In most of the underground operations, it is the current practice to handle these two benches of the bed separately at the face, and it is therefore not entirely impracticable to keep these two products separate where the mining and transportation is handled by hand, as is still the prevailing practice. Typical examples of the washing characteristics of the top bench and the lower bench of the Barro Branco coal are shown in Figs. 10 and 11.

Dona Teresa Cristina Railway: The expansion of coal production in the Santa Catarina field required substantial improvements in transportation facilities. The Dona Teresa Cristina Railway that connects the coal field with the coast was improved, and some new rolling stock was obtained, but the primary improvement that stepped up the capacity to haul coal was obtained by modernizing and greatly expanding the maintenance department so that the locomotives could be kept in service. Movement of coal also was facilitated by construction of a central railway loading bin in the railway yards at Criciuma, where raw coal may be loaded by gravity into railway cars.

Loading facilities at the twin ports of Laguna and Imbituba also were greatly improved. A loading station at the Imbituba port, erected in 1940, consists of a series of bunkers adapted to load by gravity chutes directly to ocean-going vessels. Belt conveyors distribute the coal to the bunkers and transport the coal from the railway dump-hopper to the distributing conveyors above the bunkers.

The Laguna facilities consist of an open storage yard along the waterfront with a concrete seawall and masonry quay upon which traveling cranes serve the boats alongside, loading coal from open storage behind the seawall.

Development in Parana: Exploration work was carried out in the northeast part of the State of Parana during World War II, and a small production was developed to supply fuel for the Rede Viacao Parana-Santa Catarina and the industrial area of Sao Paulo. The Parana coal beds lie in a number of small basins in the general area called the Rio de Peixe area which is served by the Rio de Peixe branch of the above-mentioned railway. The coal beds vary greatly in thickness and in quality, ranging from subbituminous to anthracite; however, the principal operating mines are in the bituminous coal. These mines are opened by drifts, but in the northeastern part of the field there are beds under about 330 ft of cover that would have to be opened by shafts.

During the development of these mines, the Rio de Peixe branch of the railway was extended to Euxebio Oliveira; it is proposed to extend that railhead to the center of the Rio de Peixe coal-producing area, but construction work is not yet completed. Inadequate transportation facilities have limited coal-production capacity. Mining and mine transportation are all carried on by hand, and preparation methods are very simple. Until 1946 practically all the coal was screened at about 1½ in., and the screenings were discarded. In 1946, the Companhia Carbonifera Brasileira installed a cleaning plant at its property and also put in a modern stoker-fired boiler plant to generate power. The preparation plant with screens, picking tables, and Deister tables washes the slack, producing a coal of around 14 to 18 pct ash content.

The peak production was reached at about 1945, when 79,856 metric tons of coal was shipped out of the Parana field. Practically all this coal was used as locomotive fuel by the Rede Viacao Parana-Santa Catarina and the Sorocabana Railway. Both these railways serve industrial areas that would be potential consumers of coal if production and transportation facilities were sufficient to make the coal available.

Chile

Chile has been a steady producer of coal for many years and has perhaps the most stable and modernly equipped coal industry in the continent. The record of production from 1938 to 1948, inclusive, is shown in Table VII. Chile is one of the few South American countries that is self-sufficient with respect to coal production. As indicated in Table VII, there is usually a small balance of exports over imports; the exported coal is usually sent to Argentina and Brazil.

The principal established coal industry is in the Arauco Bay area, where the production of coal was

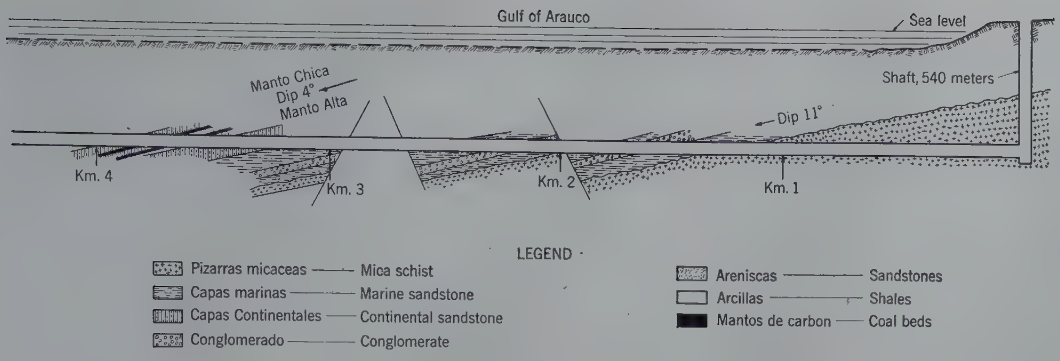

Gulf of Arauco
Sea level
Shaft, 540 meters
Manto Chica
Dip 4°
Manto Alta
Dip 11°
Km. 4
Km. 3
Km. 2
Km. 1
LEGEND
Pizarras micaceas — Mica schist
Capas marinas — Marine sandstone
Capas Continentales — Continental sandstone
Conglomerado — Conglomerate
Areniscas — Sandstones
Arcillas — Shales
Mantos de carbon — Coal beds

Fig. 12—Profile of the Piques Nuevos and gangways at Lota mine, Chile.

initiated in 1840 and has been continued intermittently since that date, with fairly steady production since 1935. The Lota mine and Schwager mine in the Arauco Bay area are the principal producers of coal in Chile. These operations were described by Toenges in a publication of the Bureau of Mines[2] reporting the findings of a technical mission sent to study the Chilean coal industry in 1944. Some illustrations in this text are taken from Toenges' report. The Lota and Schwager mines are developed by slopes and rock tunnels, and the workings in both mines extend under the sea. The method of working and relations of some of the typical workings to the coast line are shown in Fig. 12.

Metallurgical Coal: The Companhia Acero del Pacifico proposed to use coal from these mines to produce metallurgical coke and that projected development was the principal reason for the technical mission.[2] Samples of these coals, as well as some of the other Chilean coals were sent to the Central Experiment Station of the Bureau of Mines at Pittsburgh, where washing and carbonization tests were made. The Lota and Schwager coals are very low in fixed ash and sulphur and respond very readily to simple washing operations. Carbonization tests made by the Coal Carbonization Section of the Bureau indicate that these coals are moderately caking but make a rather weak, brashy coke, except when blended with low-volatile coals. Blended with 10 pct low-volatile coals, such as the Pocahontas No. 3, the Lota and Schwager coals perform similarly to the Sunnyside Utah coals in high-temperature coking. Float-and-sink test data and carbonization test data of a typical sample are shown in Tables VIII and IX.

There are other coal-mining operations in the Curanelahue, Lirguen-Cosmeto regions and in the Province of Magallanes. These have been described in detail by Toenges, and it is not necessary, therefore, to go into a descriptive treatment in this paper.

An estimate of available reserves of coal and lignite in Chile is given in Table X.

It is generally believed that there are extensive reserves of coal, as yet unexplored, in Magallanes not included in these estimates.

Colombia

The Republic of Colombia is considered to have the most extensive coal resources of any of the South American countries. Up to the present, the development of coal-producing industries has been purely local to satisfy the requirements of the internal market.

It is a peculiarity of this country that the centers of population and industry are in the interior and separated from the sea and from each other by mountain barriers that present such formidable obstacles of communication that these individual areas must be almost self-sufficient.

The most important of these industrial centers is on the great Bogota Plateau between the central and eastern mountain ranges. This is a community of miscellaneous small industries which have been expanding rapidly since 1940, and the local coal industry is almost the sole source of energy fuel. The demand is currently 40 to 50 thousand tons per month to supply the industries, the railways, and the domestic market. There is no continuous railway connection to the coast or to any of the adjacent

Table VII. Production and Movement of Coal in Chile, Metric Tons

Year	Production	Imports	Exports	Consumption
1938	1,838,000		58,000	1,780,000
1939	1,651,000		36,000	1,615,000
1940	1,740,000	221,000	32,000	1,929,000
1941	1,846,000	69,000	50,000	1,915,000
1942	1,921,000	17,000	86,000	1,852,000
1943	2,032,000	20,000	59,000	1,993,000
1944	2,279,000	16,000	46,000	2,249,000
1945	2,079,000	1,000	55,000	2,078,000
1946	1,954,000	3,000	39,000	1,912,000
1947	2,079,000	107,000	14,000	2,172,000

Table VIII. Typical Washability-Test Data of a Sample of Lota Coal

Specific Gravity Fractions	Weight, G	Weight, Pct	Ash, Pct	Sulphur, Pct	Yield of Float Coal, Pct			Cumulative Sink, Pct		
					Weight	Ash	Sulphur	Weight	Ash	Sulphur
Float, 1.35	52163.	98.52	2.28	0.71	98.52	2.28	0.71	100.00	2.72	0.73
1.35-1.40	298.	0.56	14.68	1.94	99.08	2.35	0.72	1.48	32.26	2.14
1.40-1.45	90.	0.17	16.11	1.44	99.25	2.37	0.72	0.92	42.97	2.26
1.45-1.50	57.	0.11	21.85	1.33	99.36	2.40	0.72	0.75	49.05	2.44
1.50-1.55	58.	0.11	24.50	0.99	99.47	2.42	0.72	0.64	53.73	2.64
1.55-1.58	12.	0.02	20.72	0.54	99.49	2.42	0.72	0.53	59.79	2.98
1.58-1.70	38.	0.07	36.92	0.70	99.56	2.45	0.72	0.51	61.33	3.07
Sink, 1.70	234.	0.44	65.21	3.45	100.00	2.72	0.73	0.44	65.21	3.45
	52950.	100.00								

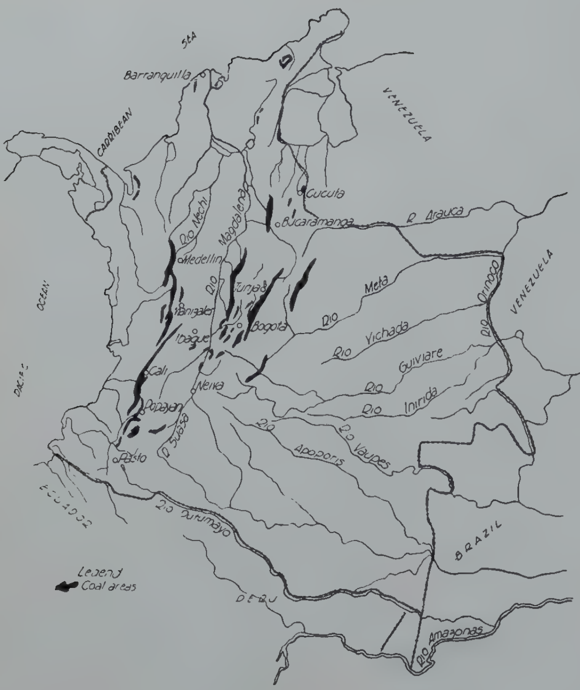


Fig. 13—Coal deposits of Colombia.

industrial valleys and practically no movement of coal in or out of the intermountain Bogota area. A very small tonnage of foundry coke is shipped from the Bogota coal field to the industrial city of Cali; this moves over the central cordillera into the Cauca Valley by truck.

There are two other smaller, but still important, industrial areas in the interior—one around the city of Cali in the Cauca Valley between the western and central ranges, the other around the city of Medellin to the north. These two centers, like Bogota, consume the entire local coal production and depend almost entirely on that source for energy fuel. Only the cities of the Caribbean coast have access to foreign sources of fuel.

Development of the national coal industry will continue to be dominated by these geographical and physiographic conditions. While each of the three major coal fields will be limited to the local market, they may expect to be substantially free from competition from outside sources of fuel. The one exception is the Cali area in the Cauca Valley. The Pacific Railway connects this coal field to the Pacific coast port of Buenaventura, and the conditions in certain respects favor the development of an export coal trade from the Cauca Valley coal field through that outlet.

Table IX. Carbonization Test of 90 Pct Schwager Mixture with 10 Pct Pocahontas No. 3. Carbonizing Temperature 900°C.

Products	Yield[a]	
	Coal, Pct	Per Ton of Coal
Coke	65.2	
Gas	18.0	11,100 cu ft
Tar	6.0	12.5 gal
Light oil	1.27	3.48 in gas
Free ammonia	0.089	18.9 lb $(NH_4)_2SO_4$

[a] Coke, tar, ammonia, and light oil are reported moisture-free; gas is reported as stripped of light oil and saturated with water vapor at 60°F and under a pressure of 30 in. of mercury.

In the 1940 to 1950 decade, the Colombian coal industry serving these three industrial areas has expanded from an annual production rate of around 200,000 tons to 1,000,000 tons. This development has followed the traditional American pioneer pattern in that it has been financed purely by local capital, under local management.

The industry is now in the hands of many small operators; production methods are very elementary; all work at the face is manual; and there are only two or three operations where even animal haulage is employed. However, the industry is in a healthy expanding condition, is supplying coal to local industries at the equivalent of $3 to $5 per metric ton, and earning a reasonable profit.

The long-term outlook is for continued gradual increase in the demand for coal; therefore, a gradual expansion of coal production is indicated. A modest modernization program, with the first step limited to small-scale mechanization of haulage and outside transport, would greatly improve the economic conditions of the industry, but a large-scale mechanized American mining project introduced without regard to local market conditions would greatly disturb the economic balance.

Coal Resources: The geographical distribution of coal deposits in Colombia is shown on the map, Fig. 13, furnished by the National Ministry of Mines and Petroleum. The coal measures, of late Cretaceous or early Tertiary age, outcrop in all the three great mountain ranges of the Andean system. The most extensive and persistent known coal field lies along the western flank of the eastern cordillera, and it is known to extend through the states of Cundinamaraca and Doyaca and beyond in a northeastern direction to the Caribbean coast. It is generally assumed that the Venezuelan coal fields west of Lake Maracaibo are of the same general series. Throughout this area the local deposits have been subjected to varying tectonic action; and, locally the nature of the coal varies from subbituminous to low-volatile bituminous. In the Cundinamaraca and Doyaca areas, there are a great number of outcropping beds varying greatly in thickness, and in dip from almost level to vertical.

In the area of the Cauca Valley, between the western and central cordillera, the coal measures apparently lie in a broad synclinal fold with the beds outcropping in the mountain ranges on both sides of the flat Cauca Valley plain. The coal measures, carrying several beds that are practically vertical in the vicinity of the city of Cali, extend from that area in a generally north and south direction through the State of Cauca, almost to the border of Ecuador.

At Cali, where there are many small operations in the Golondrinas Mountain in the outskirts of the city, the coal is of high-volatile content, is hard and lumpy, and makes a very good railway-locomotive fuel and general-purpose fuel.

South of Cali, where there are many small mining operations in the mountain range west of the valley

Table X. Estimate of Reserves of Coal and Lignite in Chile, Metric Tons

Lota and Schwager concessions	100,000,000
Arauco Province, other	100,000,000
Concepcion Province	20,000,000
Valdivia and Magallanes Provinces	5,000,000
Total	225,000,000

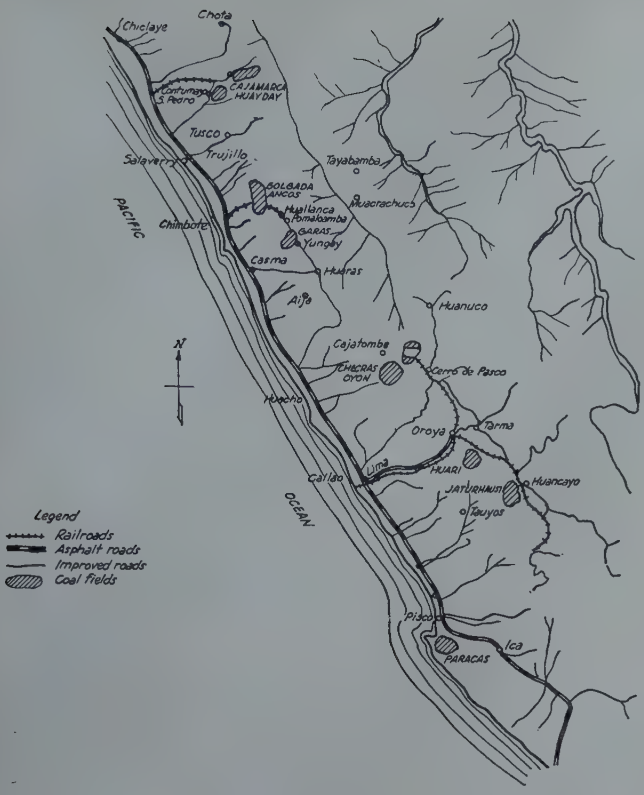


Fig. 14—Coal deposits of Peru.

and along the Popayan Branch of the Pacific Railroad, there are low-volatile deposits. Such analytical data as are available indicate that the volatile matter gradually decreases to the south, and at a point some 20 to 30 miles south of Cali there are bituminous coals of 18 to 20 pct volatile that might be adapted to the production of metallurgical coke.

In the vicinity of Medellin, the coal deposits are of lower rank and generally are relatively thick beds, level or nearly level, and better adapted to mechanical mining than the deposits in the other two areas. These coals are of subbituminous rank, but relatively low in ash and free from interbedded impurities. The product is well-adapted to local industrial use and railway locomotive fuel. There is a strong demand for fuel in that area and the local fuel-burning industry is expanding.

It is obvious from the widespread distribution of coal, as shown in Fig. 13, and the extremely mountainous nature of the terrain in the coal areas that exploration is difficult. Only a relatively few important coal-producing areas have been examined, and no reliable quantitative estimates of total national reserves are available. Tentative and preliminary estimates, based on sketchy geological reconnaissance, have placed the total at as high as 80 billion tons of recoverable coal.

Current Developments: In the Paz del Rio area of the State of Doyaca, the Government of Colombia, through the agency of the Instituto de Fomento Industrial, is currently carrying on an extensive exploration and investigative program to appraise the reserves of iron ore, coal, and limestone adapted to furnish raw materials for a small iron and steel industry projected for that region. The Koppers Co. of Pittsburgh is doing work on coal deposits. It is reliably reported that there are adequate reserves of coking coal, including low-volatile varieties adapted to blending for a suitable coking-coal mixture. In the spring of 1949, L. E. Young, consulting engineer, of Pittsburgh, Pa., made an intensive study of the coal deposits in the immediate vicinity of Paz del Rio and Belencito where the projected plant will be located and proposed a plan for the development of coal mines on the LaChapa property to produce the fuel needed for the projected metallurgical industry. Dr. Young reports that it is practicable to develop adequate producing facilities there at reasonable cost.

Nearer Bogota, in the State of Cundinamaraca, production from the coal deposits of San Jorge is being expanded to furnish the fuel needed in an alkali industry being constructed in that vicinity by the Instituto de Fomento Industrial.

In current commercial mining operations in the Bogota field, the properties of the local Lobo-Guerero family have been equipped with some mechanical aids to production; and a simple screening picking plant is being installed at one of its mines, using equipment of American manufacture, to produce domestic coal for the Bogota city market where this company has well-equipped retail yards.

Currently, a study of the possibilities of developing an export industry, based on the Cauca Valley coals, is being made by the Instituto de Fomento and the national railways of Colombia in cooperation with the U. S. Bureau of Mines, represented by a mission sent to Colombia under the terms of Public Law 402, establishing a fund for cooperative scientific work with the American Republics. This program involves a study of the characteristics of the local coals, the local market trends, and the feasibility of producing and transporting low-volatile coal to seaboard for export.

Peru

The coal situation in Peru differs from most of the other coal-producing countries in South America in that the internal demand is very small and that the capital and the principal railway are readily accessible to foreign coal through the Pacific coast port of Callao. Hence, the national coal-producing industry is definitely on a competitive basis, even in the domestic market, and must produce coal at a cost comparable to that of foreign producers. At present, the internal market is further affected

Table XI. Proximate Analysis, Sulphur, and Calorific Values of Some of the Samples of Ancos and La Galgada Anthracites

Sample No.	Description	Moisture	Volatile Matter	Fixed Carbon	Ash	Sulphur	Btu per lb[a]
112	1.70 float, J bed	4.40	3.40	87.10	5.10	0.80	13,774
212	1.70 float, G bed	4.15	3.25	82.70	9.90	0.69	13,082
312	1.70 float, I bed	3.80	3.85	84.45	7.90	0.69	13,505
412	1.70 float, H bed	3.90	7.05	82.00	7.05	0.70	13,829
512	1.70 float, F bed	4.10	4.35	84.55	7.00	0.91	14,092
612	1.70 float, ⅛ in. x 0 r.o.m.	4.75	5.35	84.25	5.65		13,081
622	1.70 float, ¼ x ⅛ in. r.o.m.	4.90	6.10	83.85	5.15		13,905
632	1.70 float, ½ x ¼ in. r.o.m.	4.60	5.70	84.95	4.75		13,995
642	1.70 float, 1 x ½ in. r.o.m.	4.67	3.55	87.00	4.78		13,799
662	Hand-picked 3 x 1½ in.	5.15	4.55	84.70	5.60		13,729
672	Hand-picked 3 in. lump	4.77	4.53	84.95	5.75		13,678
712	1.70 float, Descubierto bed	5.10	2.30	85.50	7.10		13,241
812	1.70 float, Animas bed	4.65	3.00	85.80	6.55		13,479
912	1.70 float, Balcon bed	8.50	2.65	81.95	6.90		12,811
1012	1.70 float, Esperanza bed	4.85	2.55	86.65	5.95		13,478
1112	1.70 float, Presticio bed	5.00	2.70	87.75	4.55		13,680
1412	1.70 float, Pinta bed	5.38	2.60	85.90	6.12		13,381
1512	1.70 float, Cancha bed	5.20	2.60	84.35	7.85		13,153
611	Raw ⅛ in. x 0 dust, La Galgada	3.45	6.90	57.75	31.90	0.88	10,112
621	Raw ¼ x ⅛ in., La Galgada	4.20	6.10	71.40	18.30		11,999

[a] Calculated by Goutal's formula, Calories = 82 C + aV.

Table XII. Carbonizing Tests of Oyon Coal (Yields of Carbonization Products, As-carbonized Basis)

			Yields, Weight of Coal, Pct[b]							Yields per Ton of Coal[b]				
												Light Oil, Gal		
Test No.[a]	Retort Diam, In.	Carbonizing Temperature, °C	Coke	Gas	Tar	Light Oil	Free Ammonia	Liquor	Total	Gas, Cu Ft	Tar, Gal	In Gas	Tar to 170°C	$(NH_4)_2SO_4$, Lb
279	13	900	83.7	10.2	2.3	0.43	0.099	1.7	98.4	10200	4.7	1.19	0.08	8.7
279A	13	900	68.1	15.9	5.2	0.78	0.114	7.2	97.3	10450	10.7	2.12	0.68	20.3
75	18	900	82.8	11.1	1.8	0.48	0.127	3.6	99.9	10500	3.5	1.32	0.05	11.6

[a] 279 100 pct Peruvian coal.
279A 20 pct Peruvian coal and 80 pct Sunnyside coal
75 100 pct Pocahontas No. 3 coal.

[b] Coke, tar, ammonia, and light oil are reported moisture-free; gas is reported as stripped of light oil and saturated with water vapor at 60°F and under a pressure of 30 in. of mercury.

adversely with respect to coal by a special price set-up on fuel oil under the terms of contract under which the petroleum concessions are being operated.

Currently, the national coal industry is limited to two specific enterprises, the operations of the Cerro de Pasco Copper Co. at its coal-mining property Goyllarisquisga and the anthracite mines in the Santa River Valley near the seaport of Chimbote in northern Peru.

In 1947 the production of bituminous coal amounted to approximately 140,000 tons, and the production of anthracite was 80,000 tons. The bituminous coal, practically all of which was mined at Goyllarisquisga, was used by the Cerro de Pasco Copper Co. for railway-locomotive fuel and for its smelting operations at Oroya. The anthracite production was practically all stocked at the properties of the producing mining companies; but there were some export shipments of a trial nature, and some coal was taken by coastwise vessels for bunker fuel.

The Anthracite-Producing Industry: Development of the anthracite-producing industry depends almost entirely upon the demand of foreign markets. The most promising outlet would appear to be Argentina and possibly the Far East. Some shipments have been made to both of these destinations. There is a gradually developing, but very small, domestic market principally in Lima. Currently the sole household use for coal is for cooking purposes, and the competitive fuel, as elsewhere in South America, is charcoal. Some of the producers have been active in developing small cooking stoves and grates adapted to use anthracite, and there is a large potential market but one that will be very difficult to develop.

Reserves: The principal known coal-bearing areas in Peru of practical importance are shown in Fig. 14 from an unpublished report prepared by Frank E. Wagner, for the Banco Minero del Peru. The anthracite region of northern Peru near the coastal port of Chimbote includes the Huayday and Santa River Valley fields. In the Huayday field, the anthracite mining production of the Northern Peru Mining Co., at the rate of 150 tons a day for some years, was used at the copper mines and the smelter at Quiruvilca. These beds are similar to those now being worked in the Santa River Valley area at La Galgada and at Ancos. In the Galgada-Ancos field there are a number of parallel and almost vertical veins ranging from 1 to 3 m in thickness, outcropping in the mountains between the Chuquicara and Ancos Rivers and extending south of Ancos. These beds, of Cretaceous age, are highly metamorphous and approach the graphitic stage in some areas. The volatile matter ranges from 3½ to 7 pct. Somewhat farther south, at the Mina Lord on the Santa River, the volatile ranges from 7 to 9 pct. There the bed dip is only about 20° to 30° from the horizontal.

From this northern anthracite field of Peru, the coal-bearing measures, of Cretaceous age, outcrop in a large area throughout the Andean region extending in a generally southeasterly direction across Peru and into Bolivia. There are numerous outcrops and other evidences of anthracite at many locations in the Ancos-Galgada-Limena area and extending beyond the falls of the Canon del Pato and into the upper Santa Valley where, following the outcrops of the Cretaceous farther south, the coals are of higher volatile content, changing to high-volatile bituminous in rank.

The coal deposits at Goyllarisquisga being worked by the Cerro de Pasco Copper Co. are of bituminous rank and relatively high in ash. The proven reserves are relatively small but nevertheless adequate for the immediate needs of the producing company.

In Peru, as in Colombia, the location of active coal mines is determined more by geographical and physiographic conditions than by the quality of the coal available. The Goyllarisquisga property is accessible to the Cerro de Pasco railway, and the product is adaptable to the uses of the company for locomotive fuel and general industrial purposes.

Other potential coal fields in the interior, indicated on the map, have been known in a general way for many years, and some very small operations

Table XIII. Physical Properties of Coke (Columbus Steel Corp. Methods)

						Shatter Test				Tumbler Test			
						Cumulative Pct Upon				Cumulative Pct Upon			
Test No.	Retort Diam, In.	Carbonizing Temperature, °C	True Specific Gravity	Apparent Specific Gravity	Cells, Pct	1½ In.	1 In.	¾ In.	½ In.	1½ In.	1 In.	¾ In.	½ In.
279	13	900	1.92	0.87	54.7	68.2	94.2	97.1	98.0	46.6	78.8	82.8	85.6
279A	13	900	1.93	0.81	58.0	57.6	90.2	95.0	97.5	36.0	75.1	81.8	86.1
276				0.75		43.2	75.1	83.5	92.4	20.1	55.1	65.1	75.8

279 100 pct Peruvian coal.
279A 20 pct Peruvian coal and 80 pct Sunnyside coal.
276 100 pct Sunnyside coal.

produce coal for local use. These are only indicative of the distribution of potential coal reserves throughout that general part of the country. Northeast of Lima, in the Oyon coal field, the Government of Peru through the agency of the Banco Minero, made extensive exploratory openings for several years and developed a substantial tonnage of low-volatile coal that would be readily adapted for use in the production of metallurgical coke; but these deposits are not now accessible to transportation facilities. The very rugged terrain presents formidable obstacles to railway or highway construction.

On the Paracas Peninsula south of Lima, there are thought to be extensive deposits of bituminous and subbituminous coals located favorably for the development of transport facilities that would make these coals accessible to the seacoast. However, only preliminary reconnaissance surveys have been made, and while these surveys indicate the presence of substantial reserves and some analyses show the product to have good fuel coal value, no consistent development program has been undertaken owing primarily to lack of market demand and capital resources.

Outside of the Cerro de Pasco Copper Co. operations at Goyllarisquisga, the most extensive exploration work has been done in the Galgada-Ancos-Limena anthracite field, and it may be safely estimated that approximately 1½ million tons of anthracite have been proved. Persistence of the principal beds over fairly substantial distances in the area that has been adequately explored, together with numerous observations at many points throughout the region where these beds outcrop, or are opened up by local exploratory drifts, justify the assumption that there are very large probable reserves of anthracite in that area, probably running into billions of tons; but there are no adequate exploratory data to support any definite quantitative estimates. In the nearby future there is no foreseeable market development that would justify further exploration. In the present development of the industry, the proven reserves may be considered adequate.

Technologic Development: In the Santa River Valley field, a Government-sponsored program, being carried out by the Corporacion Peruana del Santa and Banco Minero del Peru, contemplates the development of an anthracite-producing industry there in connection with a broad industrialization program that might be called the Santa River Valley project. This plan provides for eventual construction of a steel plant, cement plants, electrolytic zinc refinery and a general industrial area around Chimbote Bay, the expansion of agricultural activities in the Santa River Valley, and rehabilitation and improvement of the Santa Railway, all powered by the coal industry, and a hydroelectric power plant now under construction at Canon del Pato where the Santa River breaks through the Black Range from the upper valley to the coastal plain.

These Government agencies have been active in supporting the privately owned coal-mining industry, consisting at present of three active operations, Mis Suenos mine at La Galgada, Ancos mine on the Ancos River near Galgada, and Mina Lord at Limena on the Santa River farther south. The principal project of the Government, besides improving the transport facilities furnished by the Santa Railway, has been the construction of a modern breaker and preparation plant in the immediate vicinity of the port of Chimbote, intended to serve as a central customs separation plant to handle the output of the mines along the railroad.

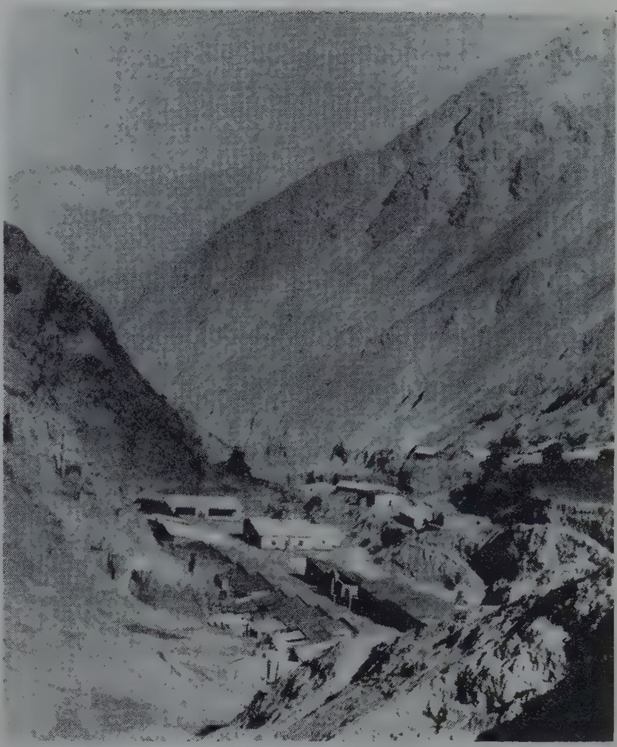


Fig. 15—Preparation plant at Ancos mine and view of Ancos river canyon.

Photograph courtesy of U. S. Bureau of Mines.

This plant, of American design and construction, provides for screening and cleaning the coal as in typical Pennsylvania anthracite practice, using Parrish-type shakers for sizing and Chance-sand flotation cones and hydrotators for washing the product. This plant only recently has been placed in operation, and a cargo of prepared coal from the plant has been shipped to the Argentine for a test. A 20,000-ton open storage bin and modern piers in the harbor of Chimbote Bay, built by the Frederick Snare Corp. for the Corporacion Peruana del Santa, will furnish loading facilities for transferring the washed product to seagoing ships in the harbor.

There are also small preparation plants at the Mis Suenos mine and at the Ancos mine, where some coal was prepared for shipment before the central washery at Chimbote was completed. Fig. 15 shows a general view of the Ancos River canyon and the washing plant of the Ancos mine. On this property the veins are vertical; they outcrop in the mountainside on each side of the canyon and are opened by drifts at approximately the valley level.

Operations at Mis Suenos mine are similar, except that the development openings are on the mountainside at perhaps 1000 ft above the level of the valley of the Chuquicara River.

At Mina Lord, at Limena on the Santa River Branch of the railway some 10 miles above the Chuquicara junction, the veins, which dip at approximately 20° from the horizontal, are opened by parallel entries driven in the coal on the strike and connected by cross entries on the dip. Before the Chimbote central washing plant was completed, coal produced was accumulated in a stock pile at the mining property. This stock pile, which aggregated

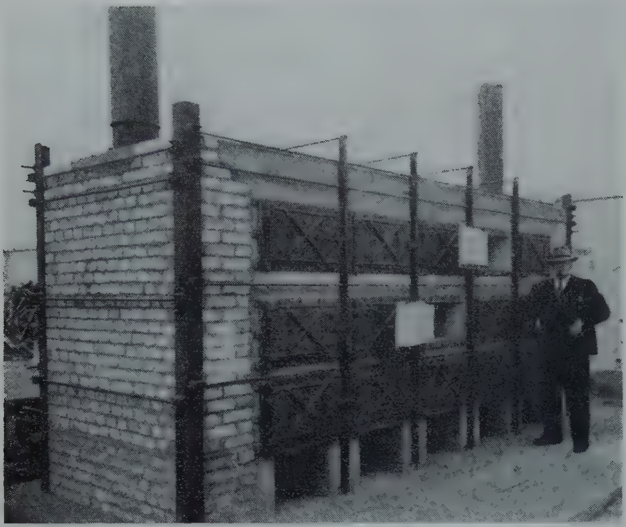


Fig. 16—Experimental coke ovens in Oyon region, Peru.

Photograph courtesy of U. S. Bureau of Mines.

approximately 80,000 tons, is now being transported to Chimbote and prepared in the central breaker there for shipment.

An investigation of the preparation characteristics of the anthracites of the Santa River region, carried out in 1945, indicated that these anthracites are readily cleanable to a good market grade and are similar in physical characteristics to our Pennsylvania anthracites. Complete data on this study of Santa River anthracites are presented in a Bureau of Mines publication.[4]

A tabulation of the principal analytical data on representative samples of the various coal beds is presented in Table XI.

In the Oyon region, an aggressive development program carried out for the Banco Minero del Peru under the supervision of Frank Wagner of Wilkes-Barre, Pa., resulted in the exploration of several workable beds of low-volatile, high-rank bituminous coal. Small-scale coking operations carried out in this region in some special retorts developed and built by the engineering organization of Branco Minero indicated that this coal is a strong coking coal adapted to produce a metallurgical fuel. Fig. 16 shows a view of this experimental carbonizing plant.

Supplementing this work, the Bureau of Mines made carbonization tests of this coal and of blends with high-volatile coals from Sunnyside, Utah. These tests indicated that the Oyon coal is a good product for such coking-coal blends. A typical test report from R. I. 4222,[5] showing the experimental data of a carbonization test of a blend of 20 pct Oyon low-volatile coal with 80 pct of the Sunnyside coal, are given in Tables XII and XIII.

Because of a lack of transportation facilities, active development work in the Oyon region has been discontinued pending financing and construction of a railway or highway that would make this region accessible to the sea through the Huara River Valley or to the Cerro de Pasco railway.

Venezuela

Venezuela is not now an important coal-producing or coal-consuming country. The very extensive and profitable petroleum industry currently overshadows any possibility of economic importance of coal. However, for the long-term future, the extensive reserves of coal probably available in that country may form the basis of an industrial development that will greatly broaden the economic structure and make the general conditions more stable.

For many years, a Government-owned coal industry in the Naricual River Valley, east of Caracas, produced coal for the use of the Venezuelan Navy. Since April 1946, when a violent explosion occurred in these mines, production has been discontinued.

At present, the most important producer of coal is the San Jose mine at Lobotera in the State of Tachira, south of the Lake Maracaibo region. This operation, producing coal for local use, has been expanding rapidly in the past three years, and in 1948 the production was estimated at some 21,000 tons.

There are outcrops and small exploration openings showing coal at many locations in western Venezuela in the Peninsula west of Lake Maracaibo and extending over the line into Colombia, and around the head of Lake Maracaibo to the south and on islands in the lake. These deposits, reportedly of Tertiary age, vary from lignitic to low-volatile bituminous, and their value for long-term future economic development is enhanced by proximity to the sea and by some evidence that these coals might be suitable for metallurgical use. Deposits are known in the six western States of Zulia, Tachira, Merida, Trujillo, Lara, and Falcon; the north-central States of Aragua and Guarico; and at least three localities in the eastern State of Anzoategui.

The Naricual coals likewise have the great advantage that they are very near to the sea and, in fact, are connected to the seaport of Guanta by a railway and an improved highway. There are also port facilities with mechanical equipment for loading coal into seagoing vessels at this Caribbean seaport, a power plant located in the railway yards at Barcelona to supply power for the railway and mining operations, and a small briquetting plant at the same location. At present, all these facilities are in an indeterminate state of repair, and some major repairs probably would be necessary to rehabilitate the general mining and transportation plant.

Although operations thus are relatively unimportant, the Government of Venezuela is very conscious of the importance of developing a coal industry or at least of an appraisal of reserves and possible economic importance of these sources of wealth. It is planned, therefore, to make a general reconnaissance of the coal fields and a study of possible uses of this product in the future development and possible industrialization of Venezuela.

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An Approximate Method of Predicting and Comparing Expected Results When Dewatering Coal by Centrifuges

by Orville R. Lyons

CENTRIFUGAL force has been utilized for the dewatering of fine coal for over 50 years by means of machines commonly called centrifugal dryers. In any centrifuge the coal and water are subjected to a spinning action which usually increases in intensity as the coal progresses through the machine. This spinning action, or the centrifugal force that it induces, tears water away from coal particles and produces a dewatered coal. The effectiveness of the dewatering action for any particular machine is governed by the size-consist of the coal and the centrifugal force imparted to the water on the coal.

Four makes of centrifuge are currently being used to dewater fine bituminous coal in the United States. These include the vertical, basket-screen C.M.I. (Centrifugal and Mechanical Industries) Carpenter, Reineveld, and the horizontal solid-bowl Bird.

The C.M.I. and Reineveld centrifuges are similar in construction and are variations of the Elmore centrifuge, varying principally in the slope of the basket, operating speed, size of perforations in the basket, and diameter of the basket. The same operating description suffices for both. These centrifuges are made up of two rotating elements, an outside conical screen frame and an inside solid cone which carries spiral pusher blades. The screen frame is supported by a hollow shaft, and the solid cone by a spindle shaft passing through the hollow shaft. Both the screen frame and the solid cone rotate at high speeds; the solid cone, with its scraping flights, rotating somewhat more slowly than the screen.

The wet coal enters the machine at the top, falls on the cone, and is thrown against the screen by centrifugal force. The coal slides down the screen until it meets the upper end of the flights, where it is moved slowly along the flights until discharged at the bottom. The effluent passes through the sieve basket or screen and discharges into a launder.

The Carpenter centrifuge consists of a conical rotating element with a vertical axis, built up of three rows of perforated stainless-steel plates or stainless-steel wedge-wire or round-wire sections cut and rolled to conform to the surface of the cone. Wet material is delivered to a conical hopper inside the centrifuge casing and is distributed in a thin, even layer over the inner surface of the cone by a disk mounted on the rotor shaft. Impacting of the coal against the screen and the centrifugal force exerted as the material moves downwards creates a dewatering effect. The moisture is forced through the coal bed and then through the perforated plates of the basket, collecting in a circular trough built into the centrifuge. The effluent is conducted from the trough through the centrifuge casing by outlet pipes. The dewatered coal is discharged over the inside bottom edge of the rotor.

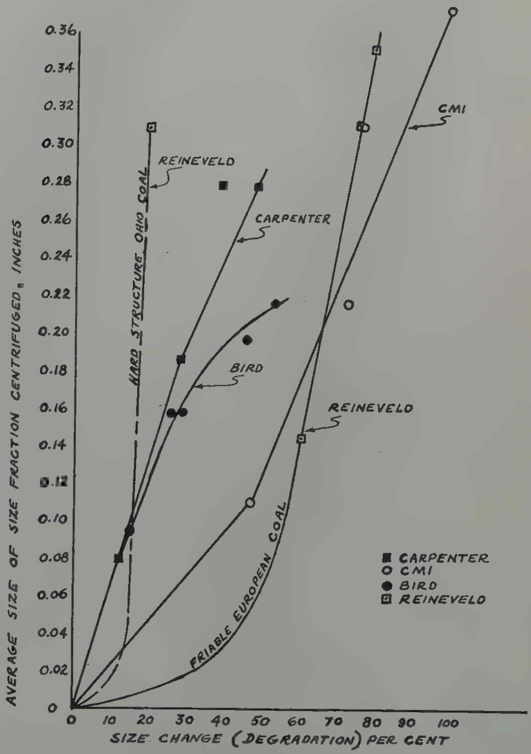

Fig. 1—Relationships existing between average size of a given size fraction and its percentage change when dewatering coal by means of centrifuges.

The Bird centrifuge consists of a tank or truncated conical shell, which is revolved at the desired speed by means of a drive sheave. A screw conveyor rotates inside the cone or *bowl* at a slightly lower speed in the same direction of rotation. The feed entrance, in the center of the large end of the truncated cone, is high enough to allow formation of a pool of slurry. Adjustable effluent-discharge ports are located in the large end of the *bowl* so the level of the liquid can be regulated. The solids are moved forward by the screw conveyor as fast as deposited and carried above the level of the pool for an interval before leaving the *bowl*.

In the past these four makes of centrifuge could be compared only by indirect means. The author published a rough comparison method in 1949[1] but

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This page of Transactions is a continuation from p. 372.
The missing pages appeared in Mining Engineering.

Table I. Carpenter Centrifuge Feed and Products Size-Consist Data

	Carpenter No. 1			Carpenter No. 2		
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
⅜x4	20.0	14.5	12.4	24.0	17.0	12.6
4x14	44.5	47.0	41.8	38.5	49.5	38.0
14x48	30.0	31.0	35.8	28.0	23.5	34.3
48x100	3.5	4.0	5.5	4.5	5.0	6.7
100x200	1.0	1.5	1.8	1.5	2.0	2.5
—200	1.0	2.0	2.7	3.5	3.0	5.9

Carpenter No. 3			
Size Mesh	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
5/16x10	70.2	55.0	50.8
10x14	8.6	20.1	19.7
14x48	14.7	17.1	17.5
48x100	5.0	5.5	7.2
—100	1.5	2.3	4.8

Carpenter No. 4	
Size Mesh[a]	Dewatered Cake Wt, Pct
¼x4	12.4
4x8	15.5
8x10	12.3
10x14	10.3
14x28	12.7
28x48	8.9
48x65	9.9
65x100	6.4
100x200	4.8
—200	6.8

Carpenter No. 5			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
8x10	7.7	9.3	2.8
10x14	12.4	15.2	7.6
14x20	21.3	23.6	17.8
20x28	17.1	19.0	19.2
28x35	14.4	14.1	14.6
35x48	7.9	6.6	7.6
48x60	3.2	2.4	2.9
60x100	6.4	3.9	6.4
100x200	4.8	2.4	4.8
—200	4.8	3.5	16.3

[a] Tyler mesh.

Table II. C.M.I. Centrifuge Feed and Products Size-Consist Data

	C.M.I. No. 1			C.M.I. No. 2
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct	Cake Wt, Pct
6x8	6.13	10.62	3.19	16.96
8x10	2.78	5.39	1.62	8.89
10x14	5.43	9.61	3.73	12.01
14x20	12.00	22.03	14.37	18.26
20x28	19.27	21.17	21.79	18.16
28x35	11.68	12.28	11.98	9.82
—35	42.71	18.90	43.32	15.90

	C.M.I. No. 3			C.M.I. No. 4		
Size Mesh[b]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
+¼	1.7	0.6	0.4			
¼x4	10.9	5.8	4.2	1.1	0.5	0.3
4x8	38.6	31.7	23.0	9.0	14.8	7.3
8x10	11.6	14.1	10.0	8.9	15.9	10.9
10x16	10.0	13.6	9.9	15.4	19.8	12.1
16x20	7.8	11.3	10.6	12.1	17.8	12.7
20x30	5.2	6.8	9.8	10.5	10.6	10.4
30x40	4.5	5.3	10.3	10.6	7.7	10.8
40x50	3.0	3.4	6.5	6.1	4.1	6.7
50x60	1.3	1.4	2.2	1.6	1.7	2.5
60x100	2.0	2.4	5.4	5.6	2.7	5.9
100x200	1.0	1.6	2.4	5.1	1.7	5.2
—200	2.4	2.0	5.4	14.0	2.7	15.2

C.M.I. No. 5			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
+¼	7.6	4.6	3.9
¼x3/16	12.4	9.7	8.3
3/16x⅛	17.6	14.8	12.6
⅛x10	18.8	18.9	16.1
10x20	15.3	18.9	16.1
20x28	6.0	8.0	6.9
28x35	5.1	6.9	6.5
35x65	7.1	8.7	10.1
65x100	2.4	2.8	4.1
100x200	2.7	3.0	5.3
—200	5.0	3.7	10.1

C.M.I. No. 6	
Size Mesh[a]	Cake Wt, Pct
6x8	4.59
8x10	13.76
10x28	52.29
28x48	16.52
48x100	6.42
100x200	2.75
—200	3.67

[a] Tyler.
[b] U.S. Standard.

Table II (Continued)

C.M.I. No. 7	
Size Mesh[a]	Cake Wt, Pct
+4	0.8
4x8	4.7
8x14	34.5
14x28	30.1
28x48	16.3
48x100	6.8
100x200	2.9
—200	3.9

C.M.I. No. 8			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
+⅜	1.4		
⅜x¼	6.9	3.0	2.6
¼x3/16	10.2	7.5	6.6
3/16x⅛	13.5	10.8	9.6
⅛x10	14.1	12.3	10.9
10x20	13.7	17.9	15.9
20x28	5.7	8.6	7.8
28x48	13.6	16.5	17.7
48x65	7.0	6.8	8.2
65x100	6.4	6.2	7.7
100x200	4.8	5.8	6.8
—200	2.7	4.6	6.2

[a] Tyler.

this comparison did not provide any direct means for determining centrifuge capacity or predicting operating results, and it was obvious that methods needed to be developed for these purposes.

Elements of Comparison

To develop a reasonably accurate method for predicting and comparing the results to be expected when dewatering coal by means of centrifuges, operating data were collected for centrifuge installations from as many sources as possible. In every case the data obtained were for actual installations tested under operating conditions. In the majority of cases this information was obtained by plant preparation engineers for their own purposes and was given to the author upon request. The major portion of the data obtained for C.M.I. and Carpenter centrifuges were obtained at installations dewatering hard or medium hard structure coals such as Illinois No. 6 or Ohio-Pittsburgh No. 8. The Bird data were obtained for both hard and friable structure coals, with the hard structure coals predominating. The Reineveld data were obtained primarily for European installations treating medium hard to very friable structure coals, but some data were obtained for an installation dewatering a hard structure Ohio coal.

The Crushing or Degradation Effect: Observations made by the author and by others, revealed that centrifuges handling very coarse coal or refuse material tended to produce a dusty cake. This condition could exist only if a major portion of the larger particles in the feed were degraded at, or just prior to, discharge from the centrifuge. Based on these observations, it seemed logical to assume that, for first approximation purposes, 100 pct of the degradation in all sizes occurred just prior to or just after discharge from the centrifuge. Later calculations show this assumption to be reasonably true.

Correlating Degradation and Particle Size: Degradation is defined readily in general, but means of measuring degradation in a preparation plant when considering a heterogeneous mixture of sizes of coal—even when the coal substance is more or less homogeneous—have not been available in the past. Studies of feed and cake size-consists have indi-

Table III. Bird Centrifuge Feed and Products Size-Consist Data

Bird No. 1			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
4x6	21.7	15.7	15.3
6x10	26.2	20.0	19.4
10x14	9.0	19.3	18.7
14x20	8.4	11.0	10.7
20x28	7.7	6.9	6.7
28x35	5.3	5.5	5.3
35x48	6.2	3.7	3.6
48x65	4.4	4.4	4.3
65x100	3.6	3.7	3.6
100x150	2.0	2.1	2.0
150x200	1.5	1.9	1.9
—200	4.0	5.8	8.5

Bird No. 2			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
6x10	26.60	22.90	22.3
10x14	16.55	15.03	14.6
14x28	19.35	19.30	18.8
28x48	6.64	9.32	9.1
48x100	5.40	7.17	7.0
100x200	5.26	5.98	5.8
—200	20.20	20.30	22.4

Bird No. 3			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
3x6	17.2	10.7	9.2
6x10	22.6	18.9	16.3
10x14	9.5	10.6	9.1
14x28	12.5	17.4	15.0
28x48	9.5	13.7	11.8
48x100	5.7	8.7	7.5
100x200	3.7	6.3	5.4
—200	19.3	13.7	25.7

Bird No. 4	
Size Mesh[a]	Cake Wt, Pct
3x6	9.8
6x10	16.6
10x14	9.7
14x28	18.1
28x48	15.5
48x100	9.6
100x200	6.4
—200	14.3

Bird No. 5			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
¼x10	50.60	46.20	37.40
10x14	10.60	10.20	8.25
14x20	6.24	7.63	6.20
20x28	4.74	6.38	5.10
28x35	3.65	5.10	4.15
35x48	2.07	3.82	3.10
48x65	1.71	3.06	2.50
65x100	1.48	2.80	2.30
100x200	2.37	4.34	3.50
—200	16.54	10.47	27.50

[a] Tyler mesh.

Table III (Continued)

Bird No. 6			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
¼x4	31.7	16.2	14.9
4x8	28.1	29.8	27.5
8x10	11.1	18.4	17.0
10x20	11.7	13.6	12.5
20x35	6.1	7.4	6.8
35x48	2.3	2.4	2.2
48x65	1.9	2.2	2.05
65x100	1.4	2.2	2.05
—100	5.7	7.8	15.0

Bird No. 7	
Size Mesh[a]	Cake Wt, Pct
3x6	9.8
6x10	16.6
10x14	9.7
14x28	18.1
28x48	15.5
48x100	9.6
100x200	6.4
—200	14.3

Bird No. 8	
Size Mesh[a]	Cake Wt, Pct
3x6	13.1
6x10	22.7
10x14	11.2
14x28	17.1
28x48	12.0
48x100	7.1
100x200	5.2
—200	11.6

Bird No. 9	
Size Mesh[a]	Cake Wt, Pct
14x28	4.4
28x35	5.7
35x48	5.9
48x65	6.2
65x100	5.9
100x200	12.4
—200	59.5

	Bird No. 10	Bird No. 11	Bird No. 12	Bird No. 13
Size Mesh[a]	Cake Wt, Pct	Cake Wt, Pct	Cake Wt, Pct	Cake Wt, Pct
4x6	8.7	2.94	17.98	19.52
6x10	37.4	19.30	24.40	23.90
10x14	10.8	12.03	9.06	10.70
14x28	16.3	20.71	15.77	15.46
28x48	10.3	14.17	10.01	9.84
48x100	4.5	10.47	6.93	6.28
100x200	3.0	5.19	3.60	3.81
—200	9.0	15.19	12.25	10.49

	Bird No. 14	Bird No. 15
Size Mesh[a]	Cake Wt, Pct	Cake Wt, Pct
¼x6	15.7	16.7
6x10	40.3	26.3
10x14	13.9	10.9
14x20	6.2	11.4
20x28	6.4	8.9
28x35	3.3	5.7
35x48	3.9	5.9
48x65	2.7	4.1
65x100	2.1	3.2
100x150	1.4	1.8
150x200	1.0	1.2
—200	3.1	3.9

[a] Tyler mesh.

Table III (Continued)

Bird No. 16		Bird No. 17	
Size Mesh[a]	Cake Wt, Pct	Size Mesh[a]	Cake Wt, Pct
35x48	4.0	4x10	43.40
48x100	14.2	10x14	13.00
100x200	21.8	14x20	8.56
200x250	15.8	20x28	6.72
250x325	27.1	28x35	5.10
—325	17.1	35x48	3.00
		48x65	2.50
		65x100	2.32
		100x200	3.44
		—200	9.26

[a] Tyler mesh.

cated that degradation was most severe in the larger sizes and apparently progressively less severe in the smaller sizes when centrifuging coal. It was concluded, after studying the operating data available, that degradation for any size fraction in a mixture of sizes could best be approximated by determining degradation for a number of closely sized fractions and plotting the results. Then degradation for intervening sizes could be read from the graph. From the operating data previously mentioned it was possible to determine the percentage differences for the top size material in a particular feed and the same size of material in the combined cake and effluent at the same installation, and then calculate the size change or degradation in per cent of the feed.

Example: ¼ x ⅛-in. coal in feed = 20 pct by weight of feed. ¼ x ⅛-in. coal in combined cake and effluent = 15 pct by weight of combined cake and effluent solids

$$\text{then degradation pct} = \frac{20-15}{20} \times 100 = 25$$

Then, from data for installations having various top sizes in the feeds, it was possible to develop for any particular make of centrifuge degradation particle-size relationships as shown in Fig. 1. The graph indicating Reineveld degradation when treating a hard structure coal represents a calculated guess based on the one point available and was included because it represents the only available data.

Size-Consist of Degraded Material: Studies of size-consist data for fine coal made over a period of years by many competent observers[2-4] reveal that the size compositions of most coals conform to a definite pattern. It therefore seems reasonable to assume that any breakage occurring in a centrifuge would tend to have essentially the same size distribution as particles of comparable size in the original feed. For example, coal in the range of ¼ x ⅛ in. would provide a degraded product having the same size-consist as the —⅛ x 0-in. coal in the centrifuge feed.

Correlating Particle Size and Surface Moisture:

Table IV. Reineveld Centrifuge Feed and Products Size-Consist Data

Reineveld No. 1			Reineveld No. 2		Reineveld No. 3			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Feed Wt, Pct	Cake Wt, Pct	Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
5/16x7	55.2	42.1	42.9	35.6	⅜x5/16	12.84	3.21	2.7
7x16	23.6	29.4	31.4	33.3	5/16x¼	5.32	3.18	2.6
16x32	9.0	10.6	12.0	13.0	¼x4	12.41	5.75	4.9
32x115	10.5	14.5	11.7	14.4	4x7	21.83	16.70	13.9
—115	1.7	3.4	2.0	3.7	7x9	19.51	21.76	18.2
					9x16	15.16	19.33	16.2
					16x32	7.20	14.33	12.2
					32x60	3.15	8.10	10.0
					—60	2.58	7.64	19.3

Reineveld No. 4			Reineveld No. 5			Reineveld No. 6			
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct
5/16x7	14.2	9.6	½x⅜	1.00	0.78	⅜x¼	0.46	0.25	0.11
7x9	7.3	6.9	⅜x4	17.18	12.10	¼x5	17.07	8.68	7.59
9x16	24.6	24.8	4x7	14.00	10.18	5x9	26.28	18.63	16.00
16x20	15.1	13.2	7x12	19.70	24.14	9x16	30.77	30.17	25.90
20x32	15.4	21.2	12x16	17.52	21.50	16x32	9.10	10.17	10.90
32x65	15.1	14.6	16x32	16.56	18.58	32x60	3.98	10.60	12.60
—65	8.3	7.1	32x65	6.78	7.32	—60	12.34	21.50	26.90
			—65	7.26	5.50				

[a] Tyler mesh.

Table IV (Continued)

Reineveld No. 7				Reineveld No. 8			Reineveld No. 9		
Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Cake Plus Effluent Wt, Pct	Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct	Size Mesh[a]	Feed Wt, Pct	Cake Wt, Pct
4x7	1.0	1.1	0.4	3x4	10.1	7.2	+ ¼	1.7	1.6
7x14	17.9	16.5	11.2	4x5	6.0	5.9	¼x3/16	6.2	5.1
14x32	44.9	44.6	35.3	5x7	13.5	11.4	3/16x⅛	21.4	19.8
32x60	25.8	20.6	28.9	7x9	24.0	25.6	⅛x10	17.3	19.0
60x100	7.9	13.0	14.7	9x16	28.1	33.3	10x20	23.2	26.3
100x150	1.3	1.6	2.7	16x32	4.5	5.0	20x28	7.7	8.4
150x200	0.4	0.8	1.6	32x48	3.8	3.5	28x48	12.8	12.2
—200	0.8	1.8	5.2	48x65	3.2	2.6	48x100	5.2	4.3
				65x150	2.1	2.0	100x200	1.9	1.4
				—150	4.7	3.5	—200	2.6	1.9

[a] Tyler mesh.

Table V. Miscellaneous Carpenter Centrifuge Data

Centrifuge No.	1	2	3	4	5
Location	Penna.	Penna.	Penna.	Illinois	Illinois
Average particle size of cake, in.	0.0939	0.0898	0.1043	0.0717	0.0312
Surface moisture content of cake, wet basis, pct	5.2	——	4.8	6.5	9.3
Average size of largest size fraction, in.	0.2800	0.2800	0.1888	——	0.079
Change in quantity of largest size fraction, pct[a]	38.0	47.4	27.7	——	12.75
Feed, tons per hr	35	55	——	——	31.2
Cake, tons per hr	30	42	——	——	18.7

[a] Degradation pct.

Test data obtained for geared-weight types of vibrating screens[5] provided a definite correlation between the average particle size of the dewatered cake and the average surface moisture content of the cake when seam moisture and inherent moisture were considered synonymous. Since this provided good correlations for vibrating screens it was assumed that the same relationship should provide correlations for centrifuges or any other type of dewatering device. Each make of centrifuge or dewatering device would be expected to have its own particular size-composition surface-moisture relationship, however. The relationships developed for centrifuges are shown in Fig. 2.

Based on the above approximations and correlations it is possible to predict, within reasonable limits, the results to be expected when dewatering fine coal by means of centrifuges and to compare the 69-in. Carpenter with the 48-in. C.M.I., the 54-in. Bird, and the "small" Reineveld. This comparison does not include the new "small" C.M.I.

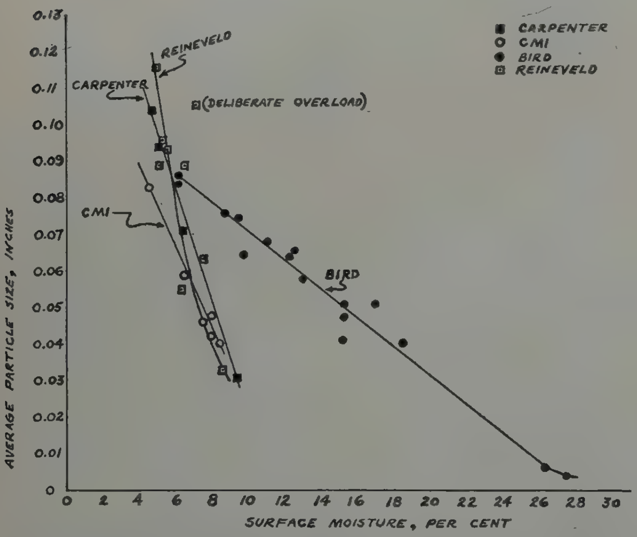

Fig. 2—Relationships existing between size of product and surface moisture content when dewatering coal by means of centrifuges.

Comparison Data

Tables I, II, III, and IV contain size-consist data for feed, cake, and combined cake and effluent solids

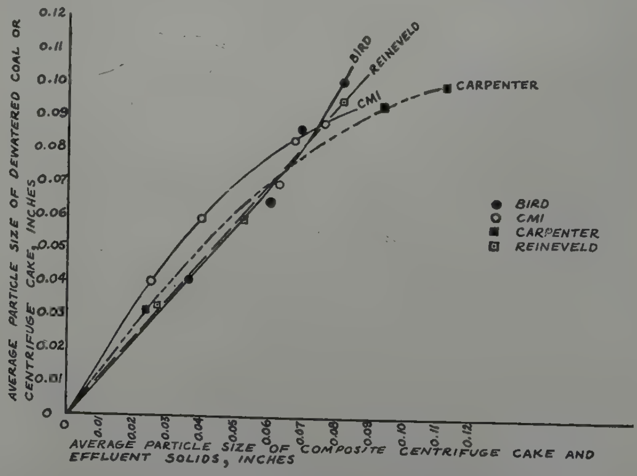

Fig. 3—Relation between average sizes of dewatered cake solids and composite cake plus effluent solids.

Table VI. Miscellaneous C.M.I. Centrifuge Data

Centrifuge No.	1	2	3	4	5	6	7	8
Location	Illinois	Illinois	Indiana	Illinois	Penna.	Ohio	Ohio	Penna.
Average particle size of cake, in.	0.04	0.048	0.083	0.0591	0.088	0.0418	0.0463	0.0698
Surface moisture content of cake, wet basis, pct	8.5	8.0	4.6	6.5		8.0	7.5	
Average size of largest size fraction, in.	0.112		0.312	0.2175	0.312			0.0380
Change in quantity of largest size fraction, pct[a]	47.9		76.4	72.6	48.6			100.0
Feed, tons per hr	50.0	50.0	58.3	35.5	60.0	34.6	44.0	50
Cake, tons per hr	15.0	12.0	42.3	17.5	51.0	21.9	30.7	44

[a] Degradation pct.

Table VII. Miscellaneous Bird Centrifuge Data

Centrifuge No.	1	2	3	4	5	6	7	8	9
Location	W. Va.	W. Va.	Penna.	Penna.	Penna.	Penna.	Penna.	Penna.	Kentucky
Average particle size of cake, in.	0.0640	0.0405	0.051	0.0514	0.0857	0.1006	0.0475	0.058	0.0064
Surface moisture content of cake, wet basis, pct	12.3	18.5	15.3	17.0	6.2		15.3	13.1	26.3
Average size of largest size fraction, in.	0.1580	0.0980	0.1970		0.1575	0.2175			
Change in quantity of largest size fraction, pct[a]	29.5	16.2	46.5		26.1	52.9			
Feed, tons per hr	35	40.0	63	50	41.2	40			
Cake, tons per hr	33.1	36.9	54	43.7	33.4	36.9			

[a] Degradation pct.

Table VII (Continued)

Centrifuge No.	10	11	12	13	14	15	16	17
Location	W. Va.	W. Va.	W. Va.	W. Va.	W. Va.	W. Va.	Penna.	Penna.
Average particle size of cake, in.	0.0644	0.0413	0.0655	0.0682	0.0842	0.0745	0.0038	0.0756
Surface moisture content of cake, dry basis, pct	9.8	15.2	12.5	11.1	6.2	9.5	27.5	8.8
Cake, tons per hr	13.5	28.0	20.0	20.0				

Table VIII. Miscellaneous Reineveld Centrifuge Data

Centrifuge No.	1	2	3	4	5	6	7	8	9
Location	Holland	Holland	Belgium	France	Holland	Hungary	Great Britain	Rhineland	Ohio
Average particle size of cake, in.	0.1158	0.1057	0.0952	0.0546	0.0935	0.0633	0.0333	0.0890	0.0826
Surface moisture content of cake, wet basis, pct	5.0	7.2	5.3	6.4	5.5	7.5	8.6	5.1	6.6
Average size of largest size fraction, in.			0.3550			0.3125	0.1475		0.3125
Change in quantity of largest size fraction, pct[a]			79			76	60		15.3
Feed, tons per hr			41.8	55	80	60	38		
Cake, tons per hr	44.0	88.0	34.7	43	60.5	54	25.3	50.5	40.0

[a] Degradation pct.

Table IX. Carpenter No. 1 Basic Degradation Data

Size Mesh[a]	Average Size, In.	Feed Wt, Pct	Degradation Pct	Cake Plus Effluent Wt, Pct
⅜x4	0.2800	20.0	48.0	12.4
4x14	0.1155	44.5	17.5	41.8
14x48	0.0288	30.0	4.5	35.8
48x100	0.0087	3.5	1.5	5.5
100x200	0.0034	1.0	0.8	1.8
—200	0.0029	1.0		2.7

[a] Tyler mesh.

Table X. Carpenter No. 1 Changing Size-Consist Relationships

Size Mesh[a]	Feed Wt, Pct	4x0 Wt, Pct	14x0 Wt, Pct	48x0 Wt, Pct	100x0 Wt, Pct	200x0 Wt, Pct
⅜x4	20.0					
4x14	44.5	55.7				
14x48	30.0	37.5	84.6			
48x100	3.5	4.4	9.8	63.6		
100x200	1.0	1.2	2.8	18.2	50.0	
—200	1.0	1.2	2.8	18.2	50.0	100.0
Pct of Feed	100.0	80.0	35.5	5.5	2.0	1.0

[a] Tyler mesh.

for Carpenter, C.M.I., Bird, and Reineveld installations. Tables V, VI, VII, and VIII contain miscellaneous data for the same centrifuges in the same order, including degradation data wherever it could be calculated for the largest size fraction in the feed. Fig. 1 presents these degradation data in graphical form including extrapolation of the data to the zero-zero point. Fig. 2 shows the relationships existing between the average particle size of the cake products and the average surface moisture content of the same cake products for the various makes of centrifuge. These data and relationships have either been discussed previously or are self explanatory.

Table IX contains basic degradation data for the feed to Carpenter No. 1. The various size fractions are presented in inches and mesh as normally presented, i.e., ⅜ in. x 4-mesh, and in terms of average particle size, in inches. The values contained in col. 4, degradation pct, were obtained from the graph shown in Fig. 1 based on the correlation between the average particle size of a closely sized fraction and degradation or size change in percent. The cake plus effluent size-consist data, col. 5, are presented for comparison with the feed size-consist data, col. 3.

Table X contains the changing size-consist relationships determined for the feed material for Carpenter No. 1 when continually decreasing the top size of the feed.

Table XI. Carpenter No. 1 Calculated Change in Particle Size Data Based on Size-Consist of Feed

Size Mesh[a]	⅜x4 Wt, Pct	4x14 Wt, Pct	14x48 Wt, Pct	48x100 Wt, Pct	100x200 Wt, Pct	—200 Wt, Pct	Totals
⅜x4	10.40						10.40
4x14	5.35	36.70					42.05
14x48	3.60	6.60	28.65				38.85
48x100	0.42	0.80	0.85	3.45			5.52
100x200	0.11	0.20	0.25	0.02	1.0		1.58
—200	0.12	0.20	0.25	0.03		1.0	1.60
Totals	20.00	44.50	30.00	3.50	1.0	1.0	100.00

[a] Tyler mesh.

Table XI contains data for the calculated changes in particle size for Carpenter No. 1. The ⅜-in. x 4-mesh size distribution shown in col. 2 was calculated as follows:

a—⅜ in. x 4-mesh in feed = 20 pct

b—Degradation of ⅜-in. x 4-mesh size = 48 pct

c—(20) (0.48) = pct of feed degraded = 9.6 pct

d—20-9.6 = 10.4 = pct of ⅜-in. x 4-mesh material left in combined cake plus effluent.

e—(9.6) (0.557) = 5.35 = pct of 4x14-mesh material in the combined cake plus effluent produced in the process of degrading the ⅜-in. x 4-mesh coal.

f—(9.6) (0.375) = 3.60 = pct of 14x48-mesh material in the combined cake plus effluent produced in the process of degrading the ⅜-in. x 4-mesh coal.

g—(9.6) (0.044) = 0.42 = pct of 48x100-mesh material in the combined cake plus effluent produced in the process of degrading the ⅜-in. x 4-mesh coal.

h—(9.6) (0.012) = 0.11 = pct of 100x200-mesh and —200-mesh in the combined cake plus effluent produced in the process of degrading the ⅜-in. x 4-mesh coal.

In the above calculations 9.6 is the pct of degraded material, (0.557) is the pct of 4x14-mesh material

Table XII. Comparison of Actual and Calculated Size-Consists for Carpenter Centrifuge Combined Cake and Effluent Solids

	Carpenter No. 1			Carpenter No. 2			Carpenter No. 3				Carpenter No. 5			
Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
⅜x4	12.4	10.40	16.2	12.6	12.48	0.9	5/16x10	50.8	50.75	0.1	8x10	6.72	6.74	0.3
4x14	41.8	42.05	0.6	38.0	37.60	3.7	10x14	19.7	13.48	46.1	10x14	12.14	11.47	5.5
14x48	35.8	38.85	8.5	34.3	36.02	5.0	14x48	17.5	24.14	37.9	14x20	21.32	20.52	3.7
48x100	5.5	5.52	0.4	6.7	6.52	2.7	48x100	7.2	8.86	23.0	20x28	19.09	17.11	9.3
100x200	1.8	1.58	12.2	2.5	2.21	11.6	—100	4.8	2.77	42.3	28x35	14.34	14.90	3.9
—200	2.7	1.60	40.7	5.9	5.17	12.4					35x48	6.94	8.33	20.0
Avg			13.2			6.05				29.9	48x60	2.56	3.42	33.6
											60x100	4.86	6.95	43.0
											100x200	3.32	5.21	57.0
											—200	8.61	5.33	61.4
											Avg			23.8

[a] Tyler mesh.

Table XIII. Comparison of Actual and Calculated Size-Consists for C.M.I. Centrifuge Combined Cake and Effluent Solids

C.M.I. No. 1			
Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
6x8	3.19	3.25	1.9
8x10	1.62	1.93	19.1
10x14	3.73	4.37	17.2
14x20	14.37	10.69	25.6
20x28	21.79	18.56	14.8
28x35	11.98	12.15	1.4
—35	43.32	49.05	13.2
Avg			13.3

	C.M.I. No. 3			C.M.I. No. 4		
Size Mesh[b]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
+¼	0.4	0.21	47.5			
¼x4	4.2	3.64	13.3	0.3	0.34	13.3
4x8	23.0	21.83	5.1	7.3	4.26	41.6
8x10	10.0	13.45	34.5	10.9	6.21	43.1
10x16	9.9	13.75	38.9	12.1	12.88	6.4
16x20	10.6	12.21	15.2	12.7	12.03	5.3
20x30	9.8	8.76	10.6	10.4	11.35	9.1
30x40	10.3	7.94	22.9	10.8	12.16	12.6
40x50	6.5	5.45	16.1	6.7	7.31	9.1
50x60	2.2	2.41	9.5	2.5	1.95	22.0
60x100	5.4	3.78	30.0	5.9	6.95	17.8
100x200	2.4	1.90	20.8	5.2	6.44	23.8
—200	5.4	4.67	13.5	15.2	18.12	19.2
			21.4			18.6

[a] Tyler mesh.
[b] U.S. Standard.

Table XIII (Continued)

C.M.I. No. 5			
Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
+¼	3.9	0.95	75.6
¼x3/16	8.3	4.68	55.6
3/16x⅛	12.6	10.73	14.9
⅛x10	16.1	17.58	9.2
10x20	16.1	19.97	24.0
20x28	6.9	9.05	31.2
28x35	6.5	8.04	23.7
35x65	10.1	11.58	14.7
65x100	4.1	4.06	1.0
100x200	5.3	4.65	12.2
—200	10.1	8.71	13.8
Avg			25.1

C.M.I. No. 8			
Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
+⅜			
⅜x¼	2.6	1.0	61.5
¼x3/16	6.6	4.01	39.3
3/16x⅛	9.6	8.13	15.3
⅛x10	10.9	12.35	13.3
10x20	15.9	16.06	1.0
20x28	7.8	7.61	2.4
28x48	17.7	19.05	7.6
48x65	8.2	10.33	26.0
65x100	7.7	9.70	26.0
100x200	6.8	7.41	9.0
—200	6.2	4.35	29.8
			21.0

[a] Tyler mesh.
[b] U.S. Standard.

in the 4-mesh x 0 fraction as shown in Table X, (0.375) is the pct of 14x48-mesh material in the 4-mesh x 0 fraction, etc. The same method was used to calculate the size distributions shown in cols. 3, 4, 5, and 6 of Table XI. Col. 7 represents the amount of —200-mesh in the original feed. The totals shown in col. 8 were obtained by adding the calculated values shown in cols. 2 through 7.

Tables XII, XIII, XIV, and XV contain actual size-consist data for combined cake plus effluent solids and the calculated size-consist data obtained by the above method of calculation for Carpenter, C.M.I., Bird, and Reineveld centrifuges. A study of the data reveals that there is a good degree of correlation between the actual and calculated size-consists.

Comparison of Centrifuges

Degradation Comparison: If one assumes the size-consist contained in col. 2 of Table XVI as being the feed material dewatered by each of the four centrifuges, then by the previous determined relationships it is possible to calculate the combined cake and effluent solids size-consist data shown in cols. 3 through 7.

By means of the data contained in Table XVI it is possible to compare the four centrifuges on the basis of degradation. The Carpenter, Bird, and Reineveld all show similar degradation characteristics when treating hard structure coals. The C.M.I. used to dewater hard structure coals shows similar degradation characteristics to the Reineveld used to dewater friable coals. For hard structure coals the C.M.I. shows the greatest amount of degradation and the other three centrifuges show lesser and almost identical amounts of degradation.

Cake Moisture Comparison: Table XVII and Fig. 3 present degradation information tabulated and plotted in a different fashion. By means of the relationships shown in Fig. 3, it is possible to determine the average size of particle for a centrifuge cake after calculating the combined cake plus effluent average size of particle. Using this relationship and the calculated data contained in Table XVI and the relationships shown in Fig. 2, the following values were obtained:

1—Carpenter cake (hard structure coal)
average particle size = 0.0605 in.
surface moisture content of cake = 7.9 pct

2—C.M.I. cake (hard structure coal)
average particle size = 0.0565 in.
surface moisture content of cake = 6.85 pct

3—Bird cake (hard structure coal)
average particle size = 0.0535 in.
surface moisture content of cake = 14.50 pct

4—Reineveld cake (hard structure coal)
average particle size = 0.0580 in.
surface moisture content of cake = 7.0 pct

On this basis the C.M.I. and Reineveld centrifuges provide cakes having the lowest surface moisture contents, the Carpenter provides a cake approximately 1 pct higher in surface moisture and the Bird provides a cake having a surface moisture content approximately double that provided by the C.M.I. and Reineveld centrifuges.

Closed System Moisture Comparison: However, it is not fair to compare the moisture content of the Bird cake with the cake moistures obtained when using the other makes of centrifuges unless some consideration is given to disposal of the various effluent solids. If one assumes that each of the four centrifuges under discussion is used to provide a dewatered cake and that the effluent solids are thickened and then dewatered by means of vacuum filters and that the centrifuge cake and vacuum filter

Table XIV. Comparison of Actual and Calculated Size-Consists for Bird Centrifuge Combined Cake and Effluent Solids

Bird No. 1				Bird No. 2				Bird No. 3			
Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
4x6	15.3	15.84	3.5	6x10	22.3	22.34	1.8	3x6	9.2	9.90	7.6
6x10	19.4	23.98	23.6	10x14	14.6	16.02	9.7	6x10	16.3	20.86	28.0
10x14	18.7	9.58	48.7	14x28	18.8	19.83	5.5	10x14	9.1	10.07	10.7
14x20	10.7	9.32	12.9	28x48	9.1	7.22	20.7	14x28	15.0	13.83	7.8
20x28	6.7	8.82	31.6	48x100	7.0	5.99	14.5	28x48	11.8	11.99	1.9
28x35	5.3	6.19	16.8	100x200	5.8	5.87	1.2	48x100	7.5	6.72	10.4
35x48	3.6	7.35	104.2	—200	22.4	22.75	1.6	100x200	5.4	4.40	18.5
48x65	4.3	5.28	22.8					—200	25.7	23.23	10.6
65x100	3.6	4.28	18.9								
100x150	2.0	2.45	22.5								
150x200	1.9	1.83	3.7								
—200	8.5	4.98	41.4								
Avg			29.2								

[a] Tyler mesh.

Table XIV (Continued)

Bird No. 5				Bird No. 6			
Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
¼x10	37.40	37.45	0.1	¼x4	14.9	14.9	0
10x14	8.25	12.42	50.7	4x8	27.5	28.56	3.9
14x20	6.20	7.65	23.4	8x10	17.0	14.18	16.6
20x28	5.10	5.94	16.4	10x20	12.5	16.09	28.7
28x35	4.15	4.65	10.3	20x35	6.8	8.94	31.5
35x48	3.10	2.66	14.2	35x48	2.2	3.46	57.3
48x65	2.50	2.20	12.0	48x65	2.05	2.88	40.5
65x100	2.30	1.88	18.2	65x100	2.05	2.15	4.9
100x200	3.50	3.10	11.4	—100	15.0	8.84	41.2
—200	27.50	22.05	19.8				
Avg			17.7				25.0

[a] Tyler mesh.

cake are combined insofar as moisture content is concerned, one will have a truer comparison of the various centrifuges from a dewatering standpoint.

Using the information provided in Tables XVI and XVII and the relationships shown graphically in Figs. 3 and 4, it is possible to calculate the average particle size of the centrifuge effluents. For example, the Carpenter combined cake plus effluent (based on the data presented in Table XVI) equals 0.0492 in. average particle size and represents 100 pct. The Carpenter cake from Fig. 3 equals 0.0605 in. average particle size and from Fig. 4 represents 72.5 pct of the combined cake plus effluent. Therefore:

$$(100)\ (0.0492) = (72.5)\ (0.0605) + (27.5)\ (x)$$

$$x = (4.92\text{-}4.39) \div 27.5 = 0.0192 \text{ in.}$$

x = average particle size of Carpenter effluent

By this method of calculation the following effluent average particle sizes were determined: Carpenter = 0.0192 in. (27.5 pct effluent); C.M.I. = 0.0214 in. (52.2 pct effluent); Bird = 0.0005 in. (10.0 pct effluent); Reineveld = 0.0162 in. (18.5 pct effluent).

The effluents from the Carpenter, C.M.I., and Reineveld will provide vacuum filter cakes ranging from 20 to 25 pct surface moisture. The Bird effluent will provide a filter cake ranging from 30 to 35 pct surface moisture. These values are based on results currently being obtained at plants treating

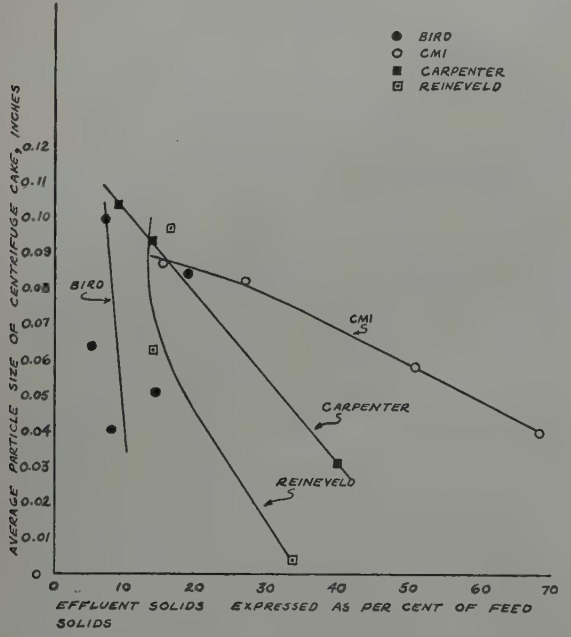

Fig. 4—Relationships existing between average size of dewatered cake solids and the percentage of feed solids reporting to the effluent when dewatering coal by means of centrifuges.

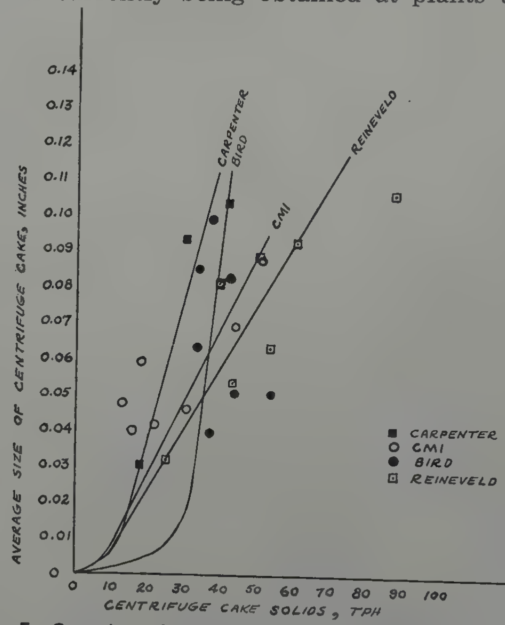

Fig. 5—Capacity relationships for centrifuges when used to dewater coal.

Table XV. Comparison of Actual and Calculated Size-Consists for Reineveld Centrifuge Combined Cake and Effluent Solids

Reineveld No. 3				Reineveld No. 6				Reineveld No. 7			
Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct	Size Mesh[a]	Actual Product Wt, Pct	Calc. Product Wt, Pct	Deviation Pct
⅜x5/16	.2.7	2.70	0	⅜x¼	0.11	0.11	0	4x7	0.4	0.4	0
5/16x¼	2.6	1.87	28.0	¼x5	7.5	4.83	35.6	7x14	10.9	8.4	22.9
¼x4	4.9	5.34	9.0	5x9	16.0	15.29	4.4	14x32	35.3	32.7	7.4
4x7	13.9	14.13	1.6	9x16	25.9	28.91	11.6	32x60	28.9	35.3	22.1
7x9	18.2	20.06	10.2	16x32	10.9	14.48	32.8	60x100	14.7	16.4	11.6
9x16	16.2	22.87	41.1	32x60	12.6	8.28	34.2	100x150	2.7	3.4	25.9
16x32	12.2	15.69	28.6	—60	26.9	28.10	4.5	150x200	1.6	1.1	31.1
32x60	10.0	8.79	12.1					—200	5.2	2.3	55.7
—60	19.3	8.55	55.7								
Avg			19.6				17.6				22.1

[a] Tyler mesh.

Table XVI. Calculated Change in Size-Consist for an Assumed Centrifuge Feed, Based on Degradation Data for Carpenter, C.M.I., Bird, and Reineveld Centrifuges

		Calculated Combined Cake and Effluent Solids				
Size Mesh	Assumed Feed Wt, Pct	Carpenter[a] Wt, Pct	C.M.I.[a] Wt, Pct	Bird[a] Wt, Pct	Reineveld[b] Wt, Pct	Reineveld[a] [c] Wt, Pct
3x6	10.1	7.07	3.53	6.06	2.9	8.5
6x8	12.3	10.87	7.36	10.64	5.8	10.5
8x14	16.5	15.72	14.00	15.81	11.0	14.7
14x28	22.3	22.98	24.07	23.40	20.4	21.2
28x48	14.6	15.87	18.13	16.14	18.6	15.6
48x100	9.2	10.31	12.15	10.48	14.2	10.6
100x200	5.5	6.25	7.51	6.36	9.2	6.7
—200	9.5	10.93	13.25	11.11	17.9	12.2
Average particle size, in.	0.0567	0.0492	0.0382	0.0472	0.0323	0.0503

[a] Hard structure coals.
[b] Friable structure coal.
[c] These figures are open to question because of the paucity of supporting data but were included because they represent at least a first approximation of the results to be expected.

Table XVII. Average Particle Size Data for Dewatered Coal or Centrifuge Cake and Composite Cake plus Effluent Solids

Centrifuge	Installation Number	Average Size of Cake, In.	Average Size of Combined Cake and Effluent, In.
Carpenter	1	0.0939	0.0938
Carpenter	3	0.1043	0.1127
Carpenter	5	0.0312	0.0241
C.M.I.	1	0.0400	0.0251
C.M.I.	3	0.0830	0.0675
C.M.I.	4	0.0591	0.0399
C.M.I.	5	0.0880	0.0761
C.M.I.	8	0.0698	0.0629
Bird	1	0.0640	0.0601
Bird	2	0.0405	0.0364
Bird	3	0.0510	0.0342
Bird	5	0.0857	0.0698
Bird	6	0.1006	0.0823
Reineveld	3	0.0952	0.0821
Reineveld	6	0.0597	0.0531
Reineveld	7	0.0333	0.0265

centrifuge effluents or comparably fine slurries by means of vacuum filters.

Then, based on the calculated surface moisture contents previously listed for the various centrifuge cakes and the vacuum filter cake moistures listed above, the following combined cake moistures were calculated. These cake moistures are based on hard structure coals. 1—Carpenter combined cake = 11.50 pct surface moisture, 2—C.M.I. combined cake = 13.70 pct surface moisture, 3—Bird combined cake = 16.55 pct surface moisture, 4—Reineveld combined cake = 9.86 pct surface moisture.

On this basis the Reineveld centrifuge, in combination with a vacuum filter, provides the lowest moisture content product. It must be remembered, however, that the Reineveld calculations for hard structure coals are based on very sketchy data.

Capacity Comparison: Fig. 5 shows the relationships existing between the average particle size of the cake and the tonnage of cake produced for the four makes of centrifuges. By means of these relationships it is possible to determine the tonnage that each make of centrifuge could handle when dewatering coal of the size-consist presented in col. 2 of Table XVI. These tonnages are as follows: Carpenter = 25.5 tons per hr of cake; C.M.I. = 34.0 tons per hr of cake; Bird = 36.0 tons per hr of cake; Reineveld = 40.0 tons per hr of cake. The Reineveld centrifuge has the greatest capacity, the C.M.I. and Bird are nearly equivalent to each other but have less capacity than the Reineveld, and the Carpenter has the lowest capacity.

Conclusions

It should be emphasized that these results are based on one set of assumed conditions. Because of the number of variables involved it would be dangerous to draw any general conclusions. Each dewatering problem will require a separate analysis and undoubtedly will lead to an individual conclusion.

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Factors Influencing the Choice of a Loading Machine

by Donald W. Mitchell

MINE operators have a choice of several classifications of mechanical loaders. Within each classification there are many types and makes available. Table I lists loaders on which manufacturing data as to operating characteristics are available. This paper discusses the conditions met in a mine as they affect these characteristics.

It is assumed that management will provide satisfactory engineering, supervision, power, maintenance, loader service within a concentration of workings, and a balanced working cycle served by balanced production equipment. These are factors which affect the optimum operational efficiency of a loader. The conditions which determine loader choice are:

Height of Vein—The main limitation to the use of a loading machine is the height to which the vein is mined; i.e. a loader cannot be higher than the place in which it is to work unless rock is taken. Since there are several low vein loaders being developed and successfully applied, this does not appear to be a requisite of an efficient mining operation.

The maximum useable height of a loader should be equal to the working seam thickness less a working clearance for travel and operation.

A—Working seam thickness =

B—Artificial roof support thickness =

C—Safe headroom below support = (generally about 6 in.)

D—Height of roadway =

E—Total of B + C + D =

A—E = maximum loader height

F—Distance from top of roadway to bottom of roof support =

G—Height of loader's coal or ore line (see Table I) =

H—Difference between F and G = Size of largest broken particle + a safety factor of 2 to 3 in.

H is a loadability factor and is important because the clearance over the top of the conveyor chain to the roof or roof support must be greater than the largest broken particle or else the particle and/or the conveyor chain may be broken. From this standpoint it might be desirable to use a lower height machine even though it may have a smaller capacity, though this is warranted only when increased lump realization offsets lower man-day and machine production rates. Ivan Given has stated, "Loading should not be handicapped by securing lump only to grind it in the breaker."

Width of Working Place—The width of a working place is the greatest width a place may be safely driven. The working place includes not only the face but the roads the loader must tram when mucking out more than one face. The width is limited by the system of placing props with sufficient room between them for the machine runner (about 2 ft).

Maximum width of place in which the loader will operate at maximum efficiency is determined by the type of mounting. With track-mounted loaders, maximum width of place is determined by maximum angular swing of the loading head and proximity of track to the face. The use of double track leads to added expense and operational complexities. Several mines that have used double-track systems have found that a substantial saving is made by the use of machines that load out wider places. Trackless loaders may work in rooms of any maximum width.

Minimum width of place in which a loader will operate is determined by width of the loader plus a safe movement area for the operator. Often a barodynamic study of the mine will show that either an increase in room width and/or a change in the system of propping may be made by using roof bolting, full or part-width room timbers, removable aluminum I-beams at the face, etc. Increasing the width of a working place will, in general, increase loader operating efficiency by providing more material per fall and by decreasing the preparation and moving time per ton. A change in the system of propping that would increase the minimum width might permit the use of a higher capacity loader with safer working conditions. Proper face timbering is necessary to insure safety from roof falls and because the speed of preparation and loading generally increases since the men work with greater assurance and there is less chance of kicking out posts.

Maximum Room Length—With the exception of the scraper and the duckbill, the limit to room

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length behind the loader is determined by the method of conveyance. Studies by R. W. Thomas and A. E. Schneider indicate that a scraper loses its efficiency when travelling distances greater than 75 to 100 ft since more time is spent conveying than is spent loading, and as the length increases, the time ratio doubles. The maximum length of a shaker pan line with a modern drive is about 400 ft. The experience of most operators has been that the addition of a loading head, swivels, bell-cranks, etc. will, by their added load to the drive, decrease this maximum length to about 250 or 300 ft.

Bottom Characteristics and Minimum Bearing Strength—The bearing strength and characteristics of the mine floor determine whether a track or trackless type loader may be used.

The scraper requires a hard bottom otherwise it will dig itself in, with the possibility of breaking the runway, increasing the load on the hoist, and contaminating the loaded material with bottom rock. Sharp humps and swags will also interfere considerably with scraper operations.

The duckbill requires a hard, relatively smooth bottom. Soft bottom makes the loading head movement sluggish, which in turn slows down or stops the movement of the material in the pan line; a hump or swag stops the head and/or takes it off the floor and away from the pile.

Almost all mine bottoms have greater bearing strengths than the minimums shown; however, as seen, the characteristics of the bottom influence the choice of mounting considerably. The use of track-mounted machines for loading and other mine operations protects soft bottom and avoids excessive contamination of the material and easily passes over humps and swags.

Material Handled and Particle Size—In the past, hard and sticky ores have relegated mechanical loaders to the shovel, shaker, and scraper types. Today, most mobile type loading machine manufacturers should be able to supply their loaders with wearing parts hardened and readily replaceable, with conveyor flights that hug the deck and sweep it clean. The abrasive characteristics of the material to be handled affects loader choice because of the maintenance problems and costs involved. Many coal operators have reported high maintenance costs caused by excessive part wearing. Since coal is not an abrasive and most of it acts like graphite and lubricates, this wearing may be attributed to the commonly associated impurities within the coal seam which are moderately abrasive and hard. When purchasing a loader for a particularly dirty seam, the coal operator, like the hard rock man, should specify that the machine be adapted to handle these abrasives successfully.

The minimum size particle handled by most loaders is dust. When mining in dusty conditions, machine specifications should provide for adequate seals to keep out the dust and to preserve efficient lubrication.

The maximum size particle handled depends upon the dipper or gathering arm dimensions, see Table I. For minimum degradation this dimension should be large in comparison with the size of particle loaded. When loading large particles, protection against overloading by slip clutches is preferred to shear pins, in the writer's experience. It is realized that these specifications would increase the loader's first cost, but they should provide lower maintenance and repair costs with higher tonnage realization for the machine.

Maximum Positive and Negative Grades—Pitches encountered in a mine will affect loader choice because one machine will not perform as well as another on certain grades. Maximum grades, as shown in Table I, may affect a mobile loader's capacity by about 25 pct, although actual tonnage mined is dependent upon the ability of the conveying unit to operate on the grade, which may result in negligible production rates.

Scrapers do not lose their capacity when operating on adverse grades when heavier hoisting and power facilities are provided. The duckbill on a positive grade almost doubles its capacity since the material will slide down the pan line; on negative grades the capacity is decreased about one third.

Side Pitch—Scrapers are not affected by side pitch when the runway is to the dip. Track-mounted loaders are not affected when level track is maintained, nor is the duckbill when the pan line is kept level or side boards are attached. The side pitch on which the other types of loaders will operate is shown in Table I; their capacity is decreased up to about 25 pct when operating on a side pitch.

Capacity—Most of the loaders in the table show manufacturers' rated capacities per minute and per 7 hr of productive working shift. Tonnage figures followed by question marks are estimations. The figures on the best tonnage reported from the field show what the machine has done. They should not be construed as being indicative of maximum capacity since few operations have as yet been developed where actual loading is more than 40 pct of the productive shift time.

To meet a loader's rated capacity, sufficient tonnage must be made available (with minimum face maneuvering), and transportation must be provided at such a rate as to remove this loaded material with minimum delay. Capacity is also dependent upon effective fragmentation and the method of laying down the broken material in a close but not tight pile.

Track vs Trackless Considerations—When determining whether to use a track or a trackless type loader, it is necessary to study the natural conditions under which the machine is expected to work.

Track-mounted and rubber-tired loaders have faster tramming speeds than the caterpillar types, and where operations are between places far apart, they are preferred.

One of the primary advantages of the trackless type loader is that it may be operated in a place of any width (greater than minimum) while track-mounted loaders, in places wider than shown in the table, require the use of double track and switches.

With track-mounted loaders the width of working face is limited to the swing of the head, loading at extreme angles is not as effective as loading directly in front of the machine. The trackless loader attacks the pile with pick-like blows rather than with the scraper or shovel effect used by the more popular types of track-mounted loaders. The trackless loader's weight is directly behind the force of the attack while the track-mounted loading mechanism is subjected to an angular force while loading or digging either side of center. In robbing and in side slabbing pillars the trackless loader has the advantage from the standpoint of spotting the conveying unit and attacking the material.

Trackless loaders are more suited to pitching areas

Table I. Loading Machine Factors

Loading Machine	Height of Loader In.	Height of Coal or Ore Line, In.	Material Handled Besides Coal	Maximum Particle Size	Min. Operating Width, Ft	Max. Operating Width, Ft	Min. Bearing Strength of Bottom	Mounting	Max. Operating Grades, Degrees
Scraper	any	any	any	varies	5	any	hard	none	any
Duckbill E Goodman	25	24	any	20x20x60 in.	4 1/6	50	hard	none	+any-5
Clarkson 24 BB	28	24	not rec.	1 ton	10	22	30-lb track	track	±17
Joy 12 BU	28¾	24		12 in.	6	any	1257 psf	cats.	±10
Goodman 660 BA	31	22		36x24x18 in.	8½	any	4000 psf	cats.	±5
Jeffrey L-500	31	25	not rec.	r.o.m.	9	27	30-lb track	track	±5
Joy 14 BU-7A	31½	25		24 in.	7½	any	2126 psf	cats.	±10
Myers-Whaley Automat low	40½	36	any	30 in.	8	24	40-lb track	track	±4
Goodman 360	41	32	any	36x24x18 in.	7½	24	40-lb track	track	±5
Joy 8 BU Low Ped.	41	35		18 in.	7	any	2360 psf	cats.	±12
Clarkson 28 FA	42	36	not rec.	1 ton	10	any	2000 psf	rubber-tired	±10
Joy 7 BU	43	37		24 in.	7½	any	4770 psf	cats.	±10
Myers-Whaley Automat No. 3	44½	38	any	30 in.	8	24	40-lb track	track	±4
Goodman 460	46	34	any	36x24x18 in.	7½	26½	40-lb track	track	±5
Jeffrey L-600	46¾	37	not rec.	r.o.m.	9	28	40-lb track	track	±5
Joy II BU	53	43		24 in.	10	any	3820 psf	cats.	±10
Sullivan Lohite	54	54	any	42 in.	5	any	30-lb track	track	any
Eimco 12 B Rockershovel	72	72	any	30 in.	5	8+	20-lb track	track	+5-30
Joy HL 3 Shovel Loader	75	75	any		3 5/6	7	20-lb track	track	±12
Gardner-Denver 9	76	76	any		5¼	7¾	20-lb track	track	+5-10
Joy 18 HR-2	84		any	24 in.	10	any	3200 psf	cats.	±10
Eimco 21 RS	85	85	any	36 in.	6	10	25-lb track	track	+5-30
Joy HL 20 SL	85		any		4 1/6	9	30-lb track	track	±12
Gardner-Denver 9H and L	79-88		any		5¼	7¾-8½	20-lb track	track	+5-10
Goodman Conway 125	84-91		any		5 2/3	14	40-lb track	track	±4
Gardner-Denver 114 and 14 H	91-97		any		6½	7¾	30-lb track	track	+5-10
Eimco 40 RS	94		any	36 in.	8	12	30-lb track	track	+5-8
Goodman Conway 50B	106-116		any	20 in.	6	14 1/3	40-lb track	track	±4
Goodman Conway 100	144		any	20 in.	6½	20	60-lb track	track	±4

because they work on steeper grades with less loss to capacity than track-mounted types. Where side pitches occur, the difficulty and expense incurred in laying out and maintaining level track is negated by the use of a trackless loader.

Where abrupt rolls occur frequently, unless good roadway is maintained, hand-loading onto conveyors appears to be the only method available to the operator today. This is one of the few conditions where hand-loading must supercede mechanization. The development of a loading mechanism to work off a chain conveyor might profitably be studied.

If the top conditions are such that close timbering and narrow entries are required, it is more difficult for the trackless than for the track-mounted loader to negotiate turns or to tram with any degree of mobility. Then too, the machine runner may be squeezed against a post and/or the post may be knocked out by a sudden side-kick. Here the use of track maintains proper and accurate man and equipment clearance.

The operators of all loaders under a height of 54 in. are a safe distance from the loading head if timbering is securely stood close to the face prior to firing the cut (see column Operating Distance to the Head in Table). Since to load out the cut the entire trackless type loader must move, a track-mounted loader might be considered safer to use under tender and heavily propped roof because only the loading head need be moved to load out the face, thus lessening the possibility of knocking out props.

In some beds trackless units working directly on the bottom might break it up, increasing contamination of the loaded material. This tendency is decreased by the use of track-mounted loaders.

In very wet and muddy sections trackless loaders have been successfully used but with large non-operating delays and with greater power consumptions. It is a matter of economics as to whether or not road maintenance will be able to pay for itself from the advantages obtained through the use of trackless units (wider rooms, higher total loading capacity, elimination of tracks and switches, etc.). Firmer traction to overcome the disadvantages of a soft bottom may be attained by "corduroying" roads (laying down straw, waste timber slabs, planking or broken refuse). Where coal is of sufficient height, about 4 in. of coal left over a bad bottom generally has made an excellent roadway for trackless equipment. But for any trackless roadway, better than good maintenance is required.

A. L. Toenges[1] has shown that with gathering haulage, rail and rubber-tired, labor costs for rubber-tired systems are less for units of the same size and traveling the same distance when track costs are included in the costs of rail-gathering systems.

The maneuverability and flexibility of the trackless loader especially adapts it to old working areas. Their use often results in substantial savings realized through the use of a natural roadway.

A saving of 10 lb or more of rail per yard may be achieved in a section using trackless type equipment with gathering motors since rail weight is then dependent upon the weight of the locomotive. Where track-mounted face equipment is used, rail weight is usually based upon the weight of the heavier loader or cutter.

With track, the cost involved in labor and supplies is not the only determining factor. When it is not used at the working places, many delays that result in scheduling the operation (conjunctive movement of drilling, cutting, conveying, gathering and loading machines) are eliminated. Because of the shorter switching interval at the face, the trackless loader

Table I. (Continued)

Loading Machine	Max. Side Pitch, Degrees	Operating Distance to Head, Ft	Rated Capacity, Tons per Min.	Max. Rated Capacity for 7 hr	Best Reported Field Shift	Power Source Used	Average Hp Demand	Repair Parts, Cost per Ton, c	Carry or Throw of Ore
Scraper	any	varies	varies	varies	130 (2y.)	dc, ac, air	varies	low	C
Duckbill E Goodman	15	varies	1	400	400	dc, ac	10	low	C
Clarkson 24 BB	5	16	8-10	4-600	750	dc, ac, air	32	2	C
Joy 12 BU	8	8	¾	315(?)		dc, ac	7	2 up	C
Goodman 660 BA		10+	4(?)	1800	1466	dc, ac			C
Jeffrey L-500	on track	10+	3	1200(?)		dc, ac			C
Joy 14 Bu-7A	8	11	5	2000(?)		dc, ac	15	2 up	C
Myers-Whaley Automat low	5	12½	3	700	766	dc, ac, air	25	2	C
Goodman 360	3½	11½	4(?)	1600	1030	dc, ac			C
Joy 8 BU Low Ped.	10	7	1½	630(?)		dc, ac, air	15	2 up	C
Clarkson 28 FA	3	15	8	5-600	650	dc, ac, air	32	1	C
Joy 7 BU	8	10	2	800(?)		dc, ac	17-22	2 up	C
Myers-Whaley Automat No. 3	5	10½	3	700	766	dc, ac, air	25	2	C
Goodman 460	3+	10½		1600	1030	dc, ac			C
Jeffrey L-600		20‡	3	1200(?)	800	dc, ac	40		C
Joy II BU	8	10	4	1600(?)		dc, ac	25-30	2 up	C
Sullivan Lohite	any	varies	varies	varies		dc, ac, air	15	low	C
Eimco 12 B Rockershovel	on track	2	20 cfm	8400 c.f.	250 tons	dc, ac, air	12	1	T
Joy HL 3 Shovel Loader	on track	2			100 tons	air	8	5	T
Gardner-Denver 9	on track	2	1		420	air	15		T
Joy 18 HR-2	8		6	2520		dc, ac	25-30		C
Eimco 21 RS	on track	2	35 cfm	14700	350 tons	dc, ac, air	20	0.8	T
Joy HL 20 SL	on track	2			150 tons	air	15	5	T
Gardner-Denver 9H and L	on track	2	1	420		air	15		T
Goodman-Conway 125	on track	4	63 cfm			dc, ac	25		C
Gardner-Denver 114 and 14 H	on track	2	2	840		air	15		T
Eimco 40 RS	on track	4	60 cfm	25200	400 tons	dc, ac, air	40		C
Goodman Conway 50B	on track	4	81 cfm			dc, ac	50		C
Goodman Conway 100	on track	4	135 cfm	2365		dc, ac			T

often has a higher total loading capacity.

The advantages of a scraper over other types of mechanical loaders are: lower first cost, lower maintenance charges, greater mobility and flexibility, and operator is a great distance from the face.

Where bottom conditions are bad, power shovels are preferred for the reasons previously mentioned. S. S. Clark states, "In areas where the broken rock is scattered (wide openings) power shovels require numerous tracks, and the time lost in switching negates the high loading capacity of these machines. In very high ground the danger of rock slides due to undercutting the broken pile of rock with the power shovel is always present. The slusher type starts cleaning out the breast, so that in a short time the drills are ready to set up, which is a definite advantage where mineable areas are limited."[2] Two manufacturers have developed caterpillar-mounted rock loaders which should find wide use in those mines that have been unable to use other types of power shovels.

A. E. Schneider has stated,[3] "Our studies indicate that up to 100 feet the capacity of the slushing machine is about that of a shaker, but beyond 100 feet the capacity of the slushing machine gradually drops due to the additional time for travel of the scraper in both directions, while the shaker has the same capacity per hour regardless of its length." R. W. Thomas[4] has estimated through experience and experimentation that 75 ft is the maximum efficient distance of haul for a scraper as compared to a shaker. This brings to notice the importance of: (1) making this comparison between a scraper and a power shovel, (2) the development of a duckbill-type loader that could be used successfully in the metal mines to replace the scraper.

Since both the scraper and the duckbill shaker require rather similar bottom and face top conditions for efficient operation, the factors influencing the choice between them are: 1—First cost (which generally favors the scraper), 2—positive grade (which generally favors the duckbill), 3—negative grade (which favors the scraper), 4—width of working place (which favors the scraper in widths greater than 50 ft), and 5—distance of travel from the face to the main conveying unit (as discussed above).

Carry or Throw of Ore—As seen in Table I, some loaders carry the ore by conveyors to the conveying unit, while others throw the ore into the unit. The main disadvantages of throwing the ore are: 1—The shock to the conveying unit which thereby limits the types of conveying units that may be used. Several mining authorities believe that a measure of the efficiency of a mining method is the ability of the method to provide for the moving of large quantities of material of large fragmented sizes with the least shock to the equipment, 2—degradation, which is primarily of interest only to the coal miner.

The main advantage of the loader that throws rather than carries the ore is that it is less expensive in both first cost and in maintenance. It has fewer moving parts and greater simplicity of design.

Power Requirements—Table I shows what type of power may be used with the various loaders. The high horsepower demand of certain loaders is not an important factor affecting their choice except in those cases where peak demand approximates average demand and power costs would be considerably increased by the use of high demand equipment.

For a long period of time, mechanical loading of coal has been accomplished largely by the use of dc machines, probably as a result of the use of old type and variable speed equipment. Most ore-mining equipment has been powered by compressed air probably because the air was already at the working face for the operation of drills, and it was assumed to be easier and cheaper to attach the air hose to the loading mechanism than to install special power cables.

Since most mines using dc power convert this from an ac source, one might raise the question as to why direct-current is used except on that equipment, such as locomotives and hoists, which requires variable speed motors. In a section, with the exception of track haulage, all loaders and auxiliary equipment might be powered easily with ac instead of dc motors, and, according to George C. Barnes, Jr.[5] definite advantages may be shown with respect to "(a) first cost, (b) maintenance, (c) total operating cost, (d) flexibility, (e) efficiency, (f) safety.

"A fair estimate of cost for a complete a-c system for a given mechanized mining section, including equipment to transform 2300 volts or more primary to useable underground voltages, circuits, motors and controls, is somewhere near 50 percent of the cost involved in installation of a comparable d-c system."

Air-driven motors have certain advantages in poorly ventilated sections of hot mines in that the exhaust air furnishes some measure of relief from uncomfortable working conditions. In gaseous mines, or in those mines where dust is combustible, compressed air has obvious advantages from the standpoint of safety.

In distant sections already furnished with compressed air and where long power lines would have to be run solely for the purpose of operating a loading machine, the air-driven motor has the advantages of lower installation costs. One of the primary objections to air-driven loaders is its utilization of power provided and needed for other purposes, this factor must be considered when deriving cost figures.

In operation, the air motor when overloaded will stall without damage, but conversely it will not provide the momentary excess power that may be needed to dig out heavy material or to overcome other obstacles. Electric motors that are not fully protected may be damaged by too frequent and prolonged overloading. Some operators who have replaced air-driven loaders with electrically operated units report lower operating and maintenance costs.

Robert C. Matson[6] has presented figures showing actual comparative costs of scraping for electric and air-operated hoists. The average shows that electric power costs per ton had a favorable ratio of about 8 to 1 as compared to similar air-operations.

It requires, theoretically, about 4 hp at the compressor to deliver 1 hp of useful work to the air-operated motor, without allowances for leakage in air lines, etc. In other words, direct electric drive will consume about 25 pct as much power to do the same amount of work as an air-drive.

Maintenance—Simplicity of design and accessibility of parts for repair and lubrication are of major importance in the choice of a loader. Unfortunately, no data comparing these points on all loaders are available to the writer. Maintenance cost figures may serve as a basis of comparison but cannot be considered reliable since the figures gathered appear to be calculated by varying methods. Loaders, such as the scraper and the duckbill, by their comparative simplicity, will be cheaper to maintain than other types of loaders. It may be reasoned that caterpillar and rubber-tired loaders should be less expensive to maintain than track-mounted units since they are of simpler construction with fewer moving parts. It is hoped that a concentrated study of maintenance problems and costs relative to the individual loaders may be made in the near future.

Availability of Parts—Some mines are located in regions that are not serviced readily by some loading machine manufacturers. The use of a loader would seriously hamper operations when fast service and delivery of normally unstocked parts is not available.

Table I—In most cases, after studying the conditions that affect the choice of a loader, the operator will have decided that one machine best meets his needs. When no decision has been made on this basis, the following points might be considered:

1—The tonnage that the loader will be required to load per shift based on the ability of the mine to provide and to remove this tonnage with minimum delay. An example: If the mining facilities (preparation at the face and outside, hoisting, etc.) are capable of taking 100 tons per hr from one loading machine section, it would be preferable to provide one loader with a 150-ton capacity rather than one which would load 300 tons per hr or two which would load 75 tons per hr.

2—If mining conditions are such that a mining method cannot be devised that would approach this condition of maximum efficiency, then the loader should be given a number of nearby places to load out.

3—As the time spent in moving from place to place increases (if the loader must wait for a place to be prepared, increase the number of places or the speed of preparation rather than decrease the loading rate), the tonnage demand on the loader is increased and a loader with its complimentary equipment of higher rated capacities is required.

4—Where mining conditions are such that the use of several different types and makes of loaders would result in high individual loader performance in sections of the mine, thought must be given to the increased complexity of providing repairs and parts, and to the difficulty of placing any loader in any part of the mine. It might be advisable for the average operator to use as few variations in makes and types as vein conditions permit.

Studies show that the maintenance cost for most loaders, mainly coal types, is about 33 pct to 50 pct the loader operating cost per ton mined. Standardization generally decreases maintenance costs, which in turn results in a decreased cost per ton mined. Although there is no reason to believe that standardization should decrease tonnage substantially, it might result in a long term increase.

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Primary and Secondary Mining with Auger Equipment

by D. M. Bondurant

Auger mining—extraction of coal with large diameter augers—is proving profitable in recovering coal left after stripping has ceased to be economical. Its many advantages—low initial cost of equipment, low operating cost, low power demand, adaptability to selective mining, simplicity of operation, etc.—have excited the interest of some in the industry in its possibilities for underground operation.

AT the present time, the coal industry is greatly interested in any method or machine that will cut the cost of producing a ton of coal, while at the same time producing a product of quality and grade suitable to the customer.

Comparatively new and, as yet, little known, auger mining is proving a profitable method of coal recovery. It may be used for the production of low-cost, high-quality coal with equal success from both the hard-to-mine seams, those with banded impurities and bad roof conditions, and the more easily mined seams as classified according to conventional mining methods.

A number of units are now in operation and production information is available that should be of particular interest to coal operators and of general interest to anyone associated with the coal-mining industry.

Auger mining refers to coal extraction with large size augers capable of drilling to great depths and having a high capacity. Consideration of the equipment and methods used in drilling an ordinary shot-hole will give a picture of auger mining on a smaller scale.

At present, auger mining is used largely for recovery of coal from the highwall after strip mining becomes uneconomical because of the type and the thickness of the overburden which must be moved.

Coal augers have many of the advantages of present continuous mining machines, along with many more. The four distinct operations of conventional mining—cutting, drilling, shooting, and loading—are eliminated and replaced by the combined operation of boring and conveying of the coal.

The coal recovery auger is an adaptation of the horizontal rock drill used for drilling shot-holes to shatter and loosen overburden in strip mining operations. During the war the Becker County Sand and Gravel Co. of Summersville, W. Va., experimented with drilling coal with 6 and 9-in. rock augers. The results led to the successive development of the 20, 24, 30 and the 36-in. machines, the latter having appeared in the coal fields within the past six months.

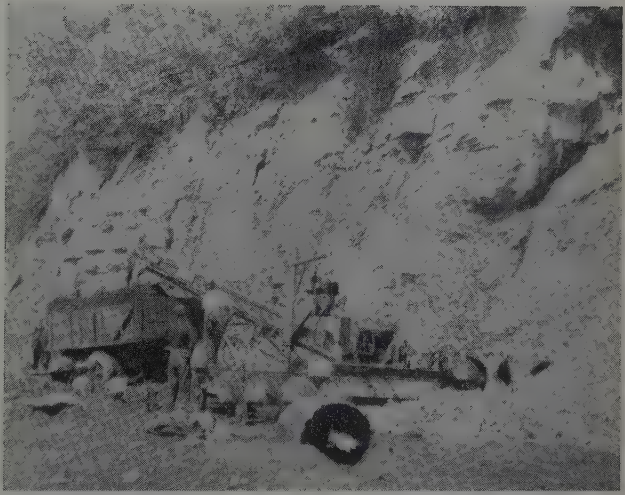

Fig. 1—Typical augering operation. Core breaker head in foreground for 30-in. auger.

Because of the great success of the smaller size

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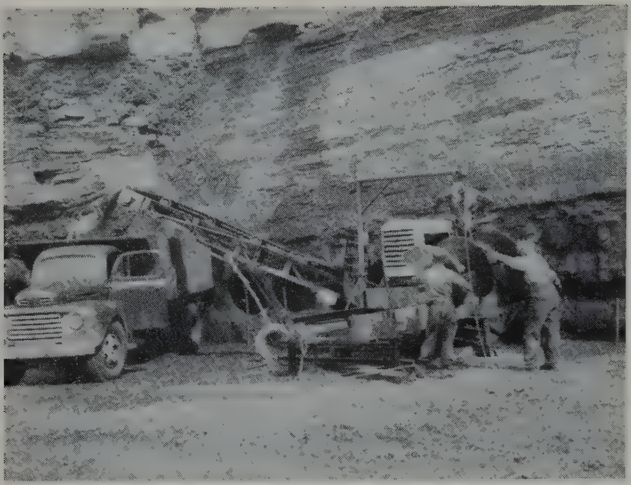

Fig. 2—Using 36-in. auger in 84-in. Eagle seam at Kingston, W. Va.

augers, it is logical to presume that larger units will be forthcoming from the commercial source.

Several manufacturers have been experimenting with and are now manufacturing machines that will bore holes up to 36 in. diam. The several machines are essentially the same except for detail construction of the drive unit, see Fig. 1.

The drill heads are of two types: the standard, and the core breaker. The standard, or slack type head, as it is called in the field, produces slack coal. It consists of a pilot cutter extending several inches ahead of three main cutter arms. Between the pilot cutter and the three main cutter arms, three stub cutter arms enlarge the hole cut by the pilot cutter. Thirty-two replaceable, chisel point, borium-tipped bits are used in each standard type head.

The core breaker head, or "lump head," consists of a steel barrel of auger diameter around which carbide tipped bits are spaced. As the barrel rotates, these bits cut a circular kerf coring out the coal. Preceding the cutting edge of the barrel by a few inches, a pilot cutter similar to the one on the standard head cuts a center hole around which the core is to be formed. Following the pilot cutter, a tapered burster follows into the center hole, breaking the core to the circular kerf, producing variable size lump—occasionally as large as the diameter of the core barrel. The position and degree of taper of the burster governs the amount and size of lump produced.

Auger sections are 6 ft long and serve to transmit power from the drive to the drilling head and at the same time convey the broken coal to the mouth of the opening.

The drive unit consists of a main frame and power unit. For recovery of coal after strip mining, the power unit generally is gasoline driven, although diesel and electric motors are optional. The power unit rotates the auger mechanically through a clutch and transmission gearing.

Forward motion is provided by a hydraulic unit which causes the power unit to travel forward and backward on the main frame a distance of approximately 8 ft. This provides ample space for attaching the next auger section. The same motor operates the hydraulic unit.

Jacks and spuds are mounted at each corner of the main frame for holding the machine against the thrust of the auger and for leveling and vertical aligning the machine.

Highwall Operation

The machine is placed at right angles to the general trend of the highwall, with the front end far enough out to clear the first auger section and drill head, see Fig. 2. The jacks are set so that the slope of the main frame, in the direction in which the hole is to be bored, coincides with the slope of the seam. Perpendicular to the line of drilling, the machine is set level. The setting of the machine with the slope of the coal is done largely by guesswork, which at times results in the boring of short-depth holes. The writer suggests that a clinometer be used to expedite setting the slope of the machine and that the back of the hole be inspected periodically for any necessary adjustment of the clinometer reading.

The coal seam, or that part of the seam being bored, should be at least 6 or 8 in. higher than the auger diameter to allow for any vertical movement of the auger, local rolls in the seam, and rotational wobble, which generally accounts for a hole diameter of 1½ to 2½ in. greater than the auger diameter.

To start operations, the drill head and one auger section are attached and the pilot cutter is started into the coal in reverse rotation to keep the cutter from wandering. As soon as the pilot cutter has a good start, the drilling is continued in second gear. The speed and feed vary with the hardness and structure of the coal. With a 24-in. auger, medium-hard coal should drill at 60 rpm with a feed of ¾ in. per revolution.

Directional control can be governed to some extent by the rate of feed. Heavy-feed pressure tends to force the head up while a light feed tends to lower it. All holes lead to the right as a result of the clockwise rotation of the auger. The lump head has better directional stability than the slack head.

After a hole has been completed, the string of auger sections is left stored in the hole and retrieved as required. Whenever a section is needed, the entire string is pulled out 6 ft by a retriever arm and chain as the drive unit is run in reverse on the frame to

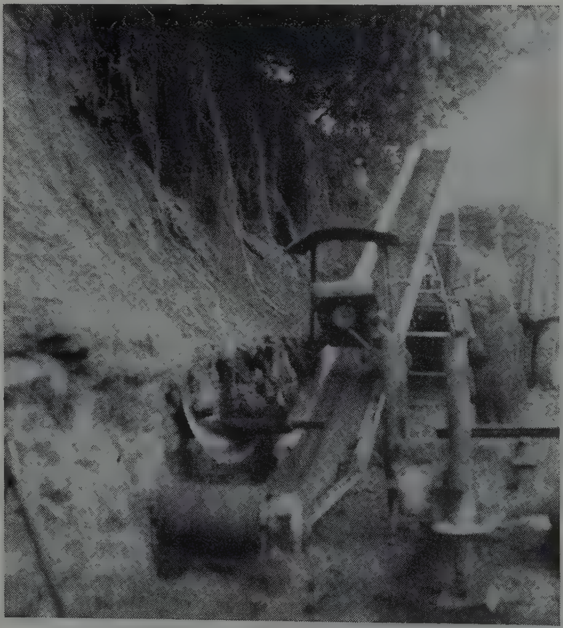

Fig. 3—Tail end drive, elevating belt conveyor, 16-in. auger in 25-in. Blue Gem seam at Barboursville, Ky.

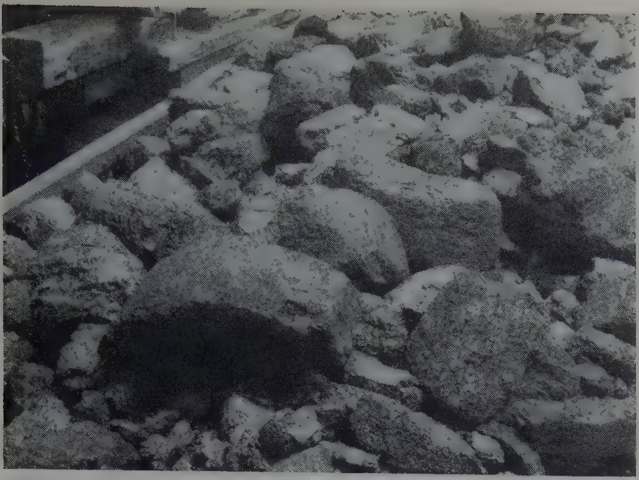

Fig. 4—Car of lump Sewell coal produced with a 30-in. core breaker head, Winona, W. Va.

permit attachment of the retrieved section. This necessitates the use of two drill heads. Time is saved, however, because rehandling of the augers is not necessary. Also, no augers are lying in the way in the limited space between the highwall and spoil.

The coal is deposited at the mouth of the opening onto a portable elevating conveyor, which conveys the coal up into a truck, as shown in Fig. 3. The conveyors are powered by small two-cylinder gasoline engines. When boring near the floor of the coal seam, it is necessary to dig out enough of the floor material to provide clearance between the tail end of the conveyor and the auger flights.

Care must be taken to bore all holes parallel. Cutting into another hole tends to bind the augers, causing breakage and bending. Alignment stakes should be set for each new setup.

The present machines are moved to new positions by skidding on greased rails. It is anticipated that future development will provide a self-propelled machine, making positioning of the unit an easy and less time-consuming job.

Hole Depth

The depth of hole is dependent on the power of the drive unit, the ability of the auger sections to resist the high torques encountered, local conditions in the seam and experience of the operator.

Since most operations at the present time are mining pillar coal left between the strip pit and old underground workings, it is hard to determine the maximum depth of holes that can be drilled. The standard length of auger proposed for the machines by one manufacturer is 60 ft. The following are maximum lengths some augers are drilling.

Auger Size, In.	Motor, Hp	Hole Depth, Ft
16	60	96
24	60	108
24	95	120
30	60	72
36	95	72

All operators have expressed their belief that they could drill to greater depths without any trouble.

Size of Coal

The size of coal is dependent on characteristic of the seam, type of head, speed of rotation and advance.

The slack head produces a product denoted as 1½-in. slack, with a small percentage up to 3-in. The sizing is fairly consistent with all size augers. A screen analysis on coal from a 30-in. slack head boring in the Clintwood seam of Virginia showed the following: 1¼ in., 27 pct; ⅜ in. x 1¼ in., 50 pct; —⅜ in., 23 pct.

Lump realization is greatly increased by using the lump head. No accurate screen analysis was available, but one company making a separation on a 2-in. screen stated that they were getting a car of lump for each car of slack. The size of the lump is surprising. It is not at all unusual to see Sewell coal drilled out in lumps as big as 10 to 24 in., see Fig. 4. The smaller sizes are largely a result of the cutting of the kerf around the core and the center hole in the core. For this reason, the use of large diameter augers will greatly decrease the percentage of smaller sizes.

Quality of Coal Mined

Selective mining of the coal seam can easily be done with coal augers. The high-quality coal of a seam can be mined without any handling of the low-quality bands or partings, thus decreasing production costs and degradation of the product, as shown in Figs. 5 and 6.

Wherever a comparison of analyses has been made between auger-mined coal and conventional or strip-mined coal, a substantial reduction in percentage of ash is noted. An operation in the Blue Gem seam of Kentucky shows an average ash of 2.1 pct for auger-mined coal, compared to 4 pct for conventional-mined coal. Another operation, in the Sewell seam, shows 3.5 pct ash for auger-mined coal and 5.0 pct for broom-swept strip coal.

Percentage of Recovery

The percentage of recovery is dependent upon the application of the proper size auger to the seam, pillar size, and ability of the operator to drill the hole as deep and as straight as possible. In most operations recovering the pillar between the strip pits and old underground workings, the coal is considered as "bonus" coal and, consequently, the percentage of recovery has not been exceptional.

The highest possible percentage of recovery by drilling the full seam height, leaving no pillar and without overlapping of the holes, is 78.5 pct, see Fig. 7.

Since it has been recommended that the auger diameter be at least 6 in. smaller than the seam or

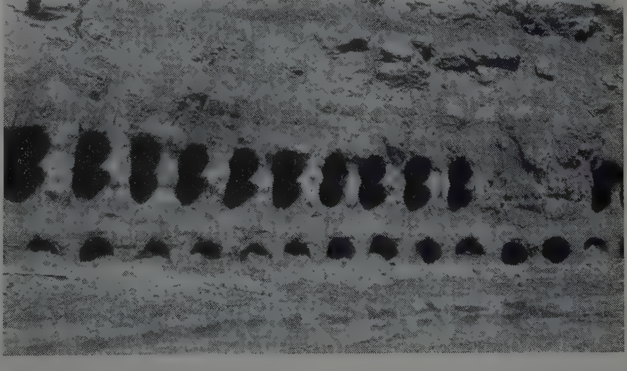

Fig. 5—Selective mining of the upper and lower Thacker seam with 30-in. auger at Thacker, W. Va. Upper seam, 7 ft; lower seam, 4 ft; separated by 2-ft "middle man."

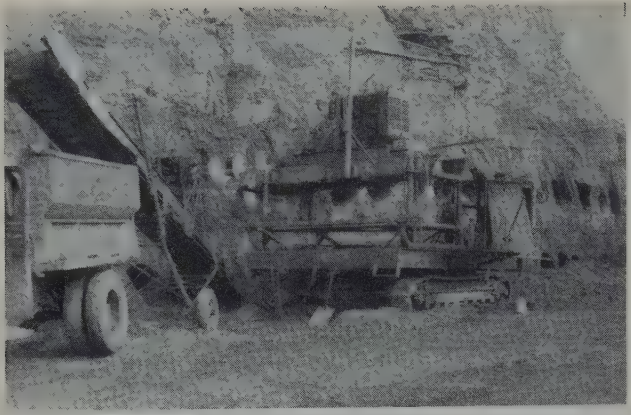

Fig. 6—Auger mounted on old concrete paver tracks for selectively mining the upper and lower Thacker seams, Thacker, W. Va.

band to be drilled, and because it will be necessary to leave some of the coal in for pillar support, the percentage of recovery necessarily will be below the maximum theoretical value.

The following is a comparison of percentage of recovery for various size augers and seam heights allowing a 6-in. pillar between holes.

Seam Height, In.	Auger Size, In.	Pillar Size, In.	Recovery, Pct
54	48	6	62.1
48	42	6	60.1
42	36	6	57.7
36	30	6	54.6
30	24	6	50.2

It is obvious from the above values that large augers in thick seams provide the greater percentage of recovery. This is because the pillar size and the allowance for vertical control of the drill is smaller in proportion to the total drilled-out area.

Very little information is available as to the proper size pillar required between holes. Most operators allow for a 6-in. pillar, mainly to decrease the chance of boring into the adjacent hole.

For a 36-in. auger, this leaves only 13 pct of the coal, measured at the smallest cross section of the pillar, to support the roof. This is a small proportion of pillar compared to the amount left in conventional room-and-pillar mining. However, the pillars offer more effective support per unit of area for several reasons: 1—the openings between pillars are comparatively small; 2—beam action is reduced because of the arching effect of the circular cut; 3—there are no shatter cracks from explosives to decrease the effective area of the pillar. For these reasons auger mining is applicable to the mining of seams with very bad roof conditions.

Proper pillar size will have to be determined by experiment. There have been instances of pillar failure at some of the operations, see Fig. 8, especially on long narrow necks of "strip islands" that have been completely undermined with augers. One company having a near mishap from pillar failure and subsequent sliding of the highwall now places a small wooden prop in the mouth of one of the holes at spaced intervals to give warning in case of roof movement.

Table I. Average Daily Output of Various Size Augers and Labor Cost at the Machine

Mine No.	Auger Size, In.	No. of Men	Tons Per Shift	Tons Per Man Shift	Labor Cost Per Ton at the Machine,[a] $
1	16	3	47	16	1.02
2	24	2	60	30	0.53
6	30	3	75	25	0.64
7	30	3	85	28	0.58
8	30	3	75	25	0.64
9	30	3	90	30	0.53
10	30	3	100	33	0.48
11	24	3	55	18	0.87
11	36	3	120	40	0.40
12	36	3	120	40	0.40

[a] At $2 per hr.

Production and Costs

A few of the companies have an additional labor cost of one or two truck drivers but many of the companies contract for haulage by the ton-mile. Supervision cost is not included.

Table I shows that additional men are not needed for operation of the larger size augers, thereby increasing tons per man shift and lowering labor cost per ton.

Gasoline consumption is slightly higher for the

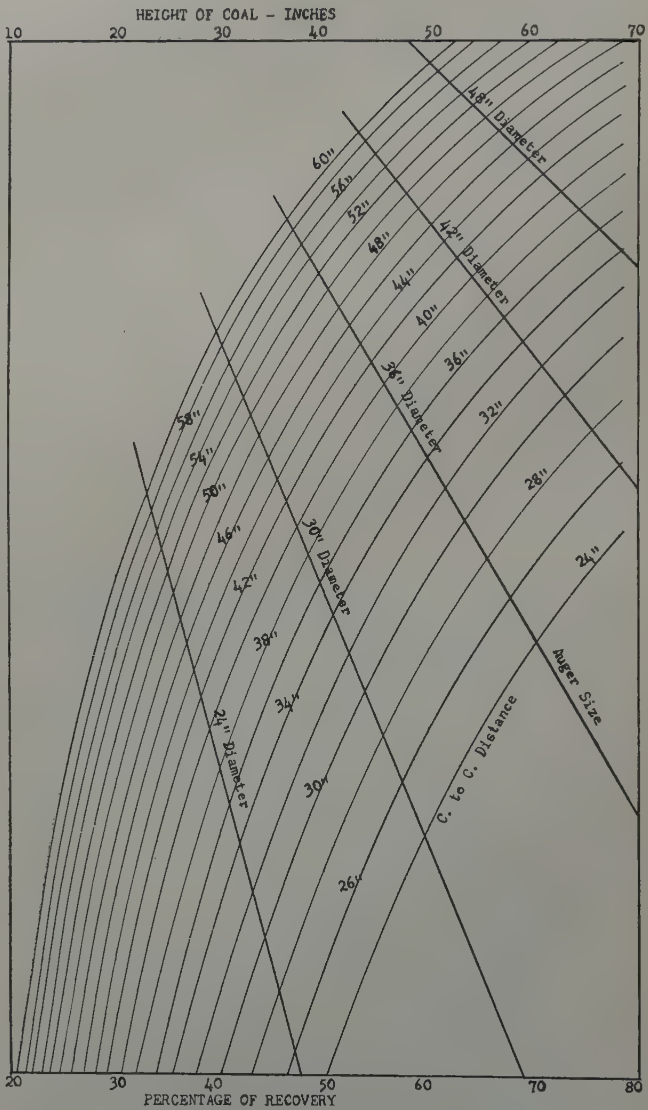

Fig. 7—Percentage of recovery for various seam heights, auger diameters, and center to center spacings.

To Use: Select coal height at top, drop vertically to auger diameter, cross horizontally to center to center spacing, then vertically to height of coal, 42 in.; auger diameter, 36 in.; center to center spacing, 42 in.; recovery, 57.7 pct.

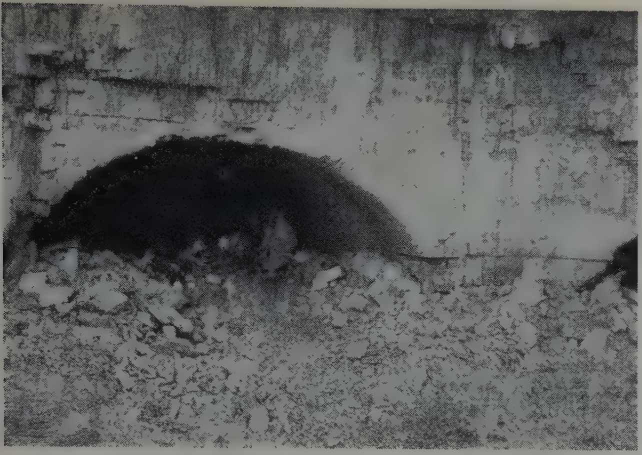

Fig. 8—Pillar failure, Eagle Seam, W. Va. Note pillar coal that has half filled 36-in. hole.

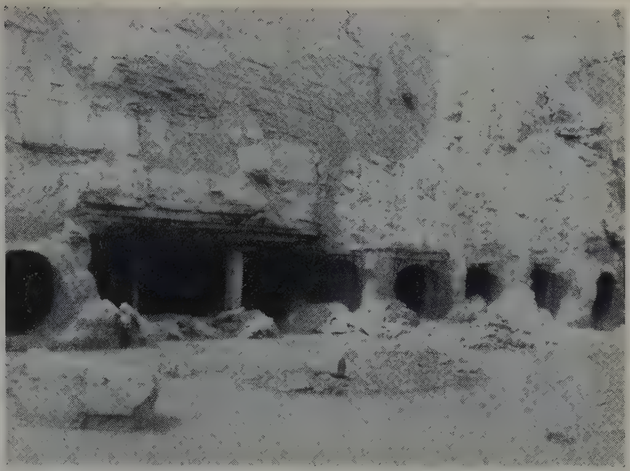

Fig. 9—Mining of pillars between holes.

larger size augers but the increased consumption is not in proportion to the increase in auger size. Twenty gallons per shift for auger and conveyor would be sufficient for any of the above operations. At a cost of $0.25 per gal, this would entail a power cost of $0.05 to $0.10 per ton. Bit cost is negligible. Most of the operators agreed that bit cost was less than $0.01 per ton.

Information is not available on depreciation and maintenance costs. It has been suggested that depreciation be figured at $0.20 per ton by basing a 50,000-ton production life on a $10,000.00 capital investment. Maintenance and repair cost has been tentatively set at the same rate. Insurance cost adds another $0.03 per ton.

A tabulation of cost per ton, excluding transportation, royalty, bulldozer, supervision, etc., shows the following: Depreciation, $0.20; repair and maintenance, $0.20; fuel for auger and conveyor, $0.10; bit cost, $0.01; labor, $0.70; insurance, $0.03; total per ton, $1.24.

These figures substantiate the fact that some operators are mining coal on a contract basis for less than $3 per ton.

Machine Operation Underground

Underground models of the coal auger have been manufactured and are being used or are awaiting developments of sections for use.

One company in particular, operating in the Cedar Grove seam of West Virginia, is now in the process of developing areas for their underground auger. The seam varies from 48 in. to 52 in. and consists of a 6-in. band of bottom coal, a 4 to 6-in. band of slate, and 38 to 40 in. of clean top coal, of which 30 in. analyzes 2½ pct ash. Roof conditions are bad. The top coal will be auger-mined with a 30-in. auger using a lump type head. The auger sections are 4 ft in length, allowing operation from a 14-ft entry.

The unit is powered with a 33-hp permissible electric motor. The machine has been used on highwalls to drill holes to a depth of 60 ft. It is anticipated that 75-ft holes and possibly 100-ft holes can be drilled without overheating of the motor.

Chain conveyors will be used for transporting the coal from the auger. Entries will be driven full seam height by conventional methods. This permits the conveyor to clear the auger very easily.

The machine crew will consist of three men—an operator to operate the machine, and two operator's helpers, who will attach auger sections, grease, pan-up, and help move. It is expected that the three men will average 100 tons per shift.

Other Applications

At an operation in Kentucky, a thin seam of coal, 25 to 28 in. thick, which has not been stripped previously, is being auger-mined from the outside. A bulldozer is used to remove enough of the outcrop coal to provide a berm wide enough for the operation of the auger. Sixteen-inch holes are being bored for a distance of 100 ft.

One operator who has two augers working on a highwall in West Virginia is contemplating the installation of an underground machine in the Pittsburgh seam to selectively mine the pillars. The seam at this mine has a bottom band of sulphur balls and a top band of low quality coal.

One manufacturer expressed the hope that the machines would prove adaptable to the mining of the steeply pitching anthracite seams. One machine is already mining coal in a steeply pitching and folded seam in Idaho.

Conclusion

Auger mining has been proved an easy and cheap method of mining high quality coal from seams of varying quality and conditions. At abandoned strip pits, coal has been auger-mined which might never have been recovered by any other method.

Auger mining holds tremendous possibilities for underground operations. There seems to be reason to believe that the cost per ton of highwall-mined coal could be duplicated underground. Entries could be driven by boring four to six holes close together and removing the thin pillars between holes, as shown in Fig. 9.

Auger mining works equally well in all types of roof conditions. Only the entries need to be timbered. Partings or impure bands in the seam can be left in place.

Four or five machines could be bought for the price of a continuous mining machine or equipment needed to outfit a working section, providing flexibility of operation and eliminating sharp cuts in daily production should a unit break down. The units are simple in design and construction and easily maintained and repaired. The power demand per unit is not high. Existing substation capacity and transmission lines would not be overtaxed. Either slack or lump coal can be mined at a steady flow that makes it possible to use existing face haulage, such as chain conveyors.

Simultaneous vs Consecutive Working of Coal Beds

by H. H. Hasler

THE mining and removal of coal from two or more beds, either simultaneously or consecutively, in vertically adjacent areas have always been matters of concern to mine operators from both operating and recovery points of view. However, with the progressive introduction and use of electrical and mechanical facilities, such problems assume even greater proportions.

In this paper typical cases occuring in Cambia County, Pa., are given and procedures whereby such problems may be greatly reduced, if not entirely eliminated, are suggested.

Throughout a considerable portion of the Central Pennsylvania bituminous coal region, particularly in Cambria County (Fig. 1), the portion of the region to which references in this paper are made, large areas are encountered where two or more beds of coal occur and have been mined extensively since the earliest days of mining in the county.

Generally, operations were commenced in one bed of coal on a property, following which, as conditions appeared to warrant, operations were commenced in another bed or beds on the same property, in some instances by the same operator.

Operating results by no means have been uniformly successful, as the following review of some typical cases indicates, partly because of lack of experience in earlier cases and in certain later cases to disregard for the experiences of earlier operators. It is suggested that repetitions of such failures may be reduced largely, if not eliminated entirely, by adherence to a few basic principles founded upon past experiences.

Geology and Stratigraphy

The geology, stratigraphy, and relief of almost all of Cambria County have been studied and reported at length by the U. S. Geological Survey in its Johnstown, Ebensburg and Barnesboro-Patton Folios and other publications, by the Pennsylvania Geological Survey and by various individuals.

All exposed rocks occurring within the boundaries of the county are of sedimentary origin, consisting of alternate beds of sandstone, slate, shale, fireclay, coal, limestone, etc., and all have been classified by the U. S. Geological Survey as extending from the upper strata of the Catskill formation, Devonian system, to and including, in a few hilltop areas, the lower strata of the Monongahela formation, Carboniferous system.

Included within that range is the Allegheny formation, Pennsylvania series, which embraces all of the commercially important beds of coal that occur in the county, viz., the *A*, or Brookville, to the *E*, or Upper Freeport, both inclusive, shown in Fig. 2.

The measures are traversed by a series of anticlines and synclines, extending in a northeasterly and southwesterly direction, which form broad, gently sloping basins (flat to 15 pct pitch, exceptionally to 25 pct). However, the topography of the surface is such that long lines of outcrops of all beds of coal overlying bed *B*, and to a limited extent bed *B* also, occur over large areas, a factor which contributed to the early and extensive development of mining activities in the county.

Earliest mining developments of commercial importance were made in *B*, *C'*, *D*, and *E* beds of coal, and it is in these beds that the principal mining operations are being conducted. To a limited extent, mining operations also have been conducted in local areas in the *A*, *B* rider and *C* beds.

Coal beds being mined today range in thickness from approximately 28 to 60 in. Intervals between the respective beds of coal are variable but conform somewhat to the following general averages: *A* to *B*, 60 to 100 ft.; *B* to *C*, 50 ft.; *C* to *C'*, 70 ft.; *C'* to *D*, 45 ft.; and *D* to *E*, 45 ft.

The following are typical mining cases observed over a period of years in various parts of the county.

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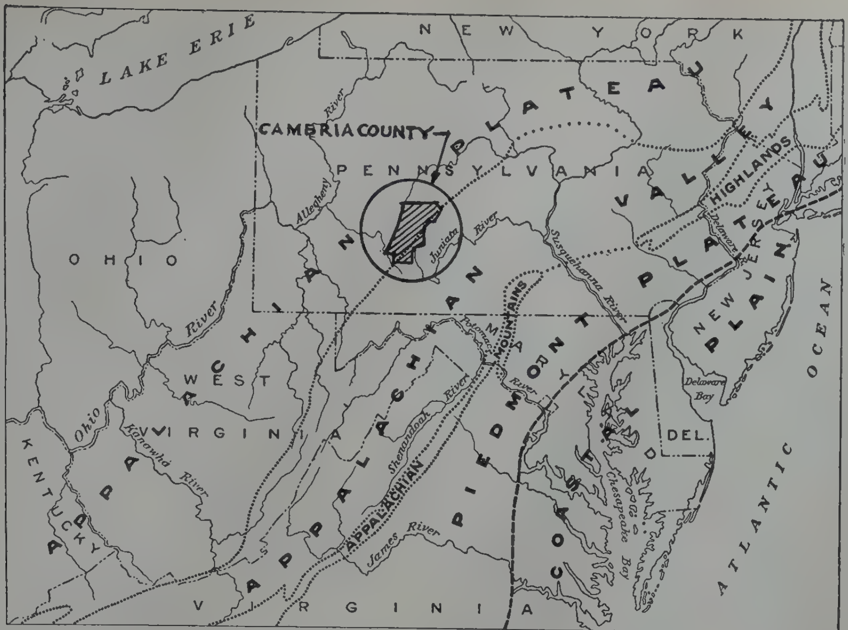

Fig. 1—Central Pennsylvania bituminous coal region.

CASE NO. 1.

1. **Bed *B*, average 48 in.**
2. **Bed *B*-rider, average 40 in.**
3. **Interval between beds, approximately 18 ft.**
4. **Cover over bed *B*-rider, approximately 100 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) **The mines were operated** by different operators and no coordination existed between plans of mining or operations in the two mines.

(b) **Bed *B* was developed** about 15 years before bed *B*-rider. Robbing of pillars in bed *B*-rider caused roof breakage and crushing of the pillars along the main haulageway in bed *B* and threatened to close the latter mine. Robbing of pillars in bed *B*-rider workings immediately above critical areas in bed *B* workings was discontinued.

CASE NO. 2.

1. **Bed *B*, average 40 in.**
2. **Bed *D*, average 42 in.**
3. **Interval between beds, approximately 170 ft.**
4. **Cover over bed *D*, approximately 300 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) **The mines were operated** by different operators and no coordination existed between plans of mining or operations in the two mines.

(b) **Bed *D* was first** developed throughout approximately 600 acres, the greater portion of the property. Bed *B* was opened and developed about 15 years later.

Removal of pillars in bed *B* mine caused crushing and squeezing of large entry pillars in bed *D* and ultimate abandonment of a considerable portion of the latter property.

CASE NO. 3.

1. **Bed *B*, average 38 in.**
2. **Bed *D*, average 45 in.**
3. **Interval between beds, approximately 160 ft.**
4. **Cover over bed *D*, approximately 400 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) **Different operators control** the mines and there is no coordination between plans of mining or operations in the two mines.

(b) **Robbing of pillars** in bed *B* resulted in marked, but as yet not fully determinable, damage to the immediately overlying active workings in bed *D*. Principal damage resulted to strata overlying bed *D*, causing caves and breaking of timbers. In certain areas the timbers were replaced as often as four times. Lateral movements of coal and overlying strata were also clearly discernible in bed *D* areas, which had been rock-dusted just prior to the disturbances.

CASE NO. 4.

1. **Bed *B*, average 44 in.**
2. **Bed *D*, average 42 in.**
3. **Interval between beds, approximately 145 ft.**
4. **Cover over bed *D*, approximately 300 to 400 ft.**
5. **System of mining, room and pillar in bed *B*; room and pillar and V-system in bed *D*.**
6. **General observations:**

(a) **The mines are operated** by the same operator but no coordination exists between plans of mining or operations in the two mines.

(b) **With few exceptions**, the mining in bed *B* preceded by many years the mining in bed *D* and, except for a few minor breaks which were observed at the bottom of bed *D*, no appreciable effect of such prior mining in bed *B* was observed.

(c) **More recently, however,** a case was noted where pillars were being extracted in bed *B* mine directly under active workings in bed *D* mine. Caving developed simultaneously in a robbed area in bed *B* mine and in the immediately overlying workings in bed *D* mine and required temporary suspension of operations in the affected portion of the latter mine. Robbing of pillars in bed *B* mine in areas directly under critical areas in bed *D* mine was immediately discontinued.

CASE NO. 5.

1. **Bed *B*, average 28 in.**
2. **Bed *E*, average 46 in. (plus 10 in. bony on top).**
3. **Interval between beds, approximately 220 ft.**
4. **Cover over bed *E*, approximately 300 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) **The mines were operated** by the same operator but no coordination existed between plans of mining or operations in the two mines.

(b) **Mining in bed *B*** was conducted approximately 20 to 30 years after mining in bed *E*. No effects of mining in either bed were found in the other.

CASE NO. 6.

(Reported to but not confirmed by the writer)

1. **Bed *B*, average 42 in.**
2. **Bed *E*, average 42 in. (plus 8 in. bony on top).**
3. **Interval between beds, approximately 180 ft.**
4. **Cover over bed *E*, 600 to 850 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) **Main entry pillars** are columnized and mining operations in the two beds of coal are coordinated.

(b) **Plan of operations** provides for mining bed *B* first, allowing two years for subsistence, and then mining bed *E*. This was first tried with a six-months to one-year interval for subsistence, but when trouble was encountered in bed *E*, the two-year period was decided upon.

(c) **When this order** of mining is followed, bed *E* is not appreciably affected except for marked reduction in the amount of water encountered in the mining of that bed, indicating the presence of fractures through which the water percolates to lower strata. Recovery of coal is normal from both beds.

(d) **Marked disturbances** occurred in bed *E* workings where immediately underlying bed *B* pillars had been extracted.

CASE NO. 7.

1. **Bed *D*, average 48 in.**
2. **Bed *E*, average 40 in.**
3. **Interval between beds, approximately 43 ft.**
4. **Cover over bed *E*, approximately 200 to 300 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) **The mines were operated** by different operators and there was no coordination between plans of mining or operations.

(b) **The property here** represented was rather small, comprising approximately 300 acres, and the *D* bed of coal was developed over the greater portion of the property about 11 years before operations were begun in bed *E*.

Shortly thereafter, the operator of bed *E* mine claimed that mining out of the bed *D* coal was causing damage to his operations and secured a permanent injunction against the operator of bed *D* mine, which restrained him from causing further damage to bed *E* operations. Consequently, large areas of bed *D* coal were lost.

CASE NO. 8.

1. **Bed *D*, average 51 in.**
2. **Bed *E*, average 41 in.**
3. **Interval between beds approximately 42 ft.**

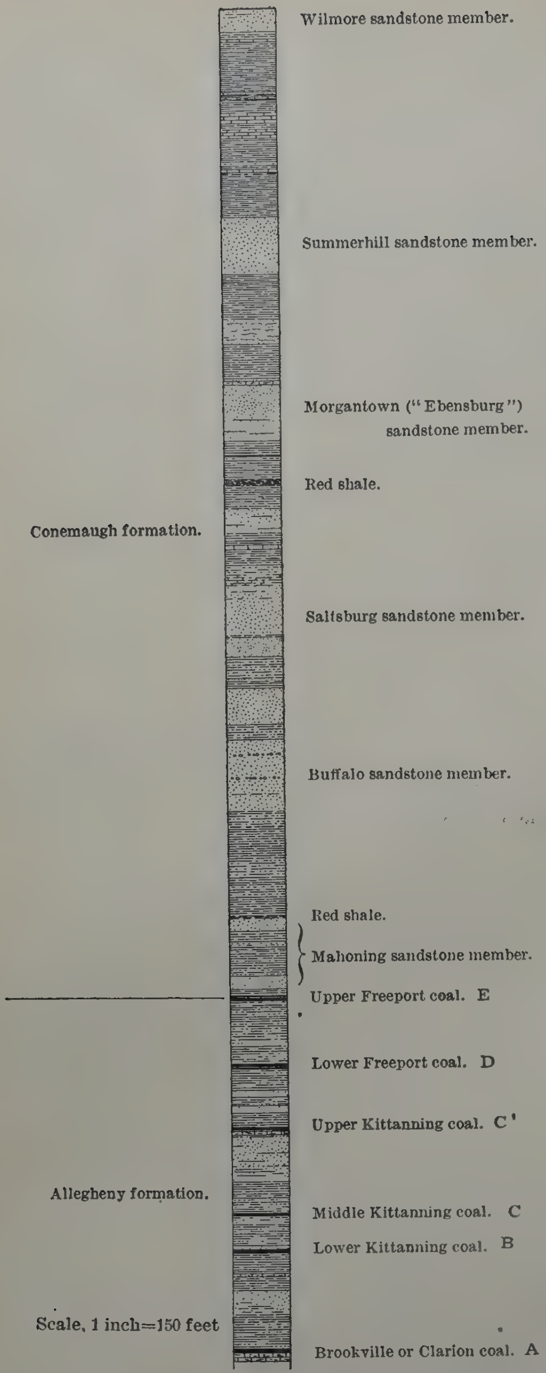
Wilmore sandstone member.
Summerhill sandstone member.
Morgantown ("Ebensburg") sandstone member.
Red shale.
Conemaugh formation.
Saltsburg sandstone member.
Buffalo sandstone member.
Red shale.
Mahoning sandstone member.
Upper Freeport coal. E
Lower Freeport coal. D
Upper Kittanning coal. C'
Allegheny formation.
Middle Kittanning coal. C
Lower Kittanning coal. B
Scale, 1 inch=150 feet
Brookville or Clarion coal. A

Fig. 2—Composite diamond-drill and surface section showing the character of the Conemaugh and Allegheny formations in the Johnstown quadrangle.

U. S. Geological Survey, Johnstown Folio No. 174

4. **Cover over bed *D*, approximately 150 to 200 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) Ownership and operation of the coal and the mines in different parts of the property varied. In certain parts, beds *D* and *E* were owned and operated by the same operator; in other parts, one operator owned and operated bed *E* only, while bed *D* was owned and operated at the same time by another operator.

(b) To the extent that beds *D* and *E* were owned and operated by the same operator, the following order of mining was observed:

Bed *D* mine was opened about 12 years before operations began in bed *E*, by which time a considerable portion of Bed *D* property had been mined over but only partially exhausted.

While full retreat operations were being conducted in the more remote sections of bed *D* mine, development work progressed in outlying areas in bed *E* mine.

Main entries in bed *E* mine were projected and driven, with pillars columnized over main entries in bed *D* mine to areas where bed D coal had been completely exhausted. Then they were driven on independent projections.

Pillar extractions in bed *D* mine were unusually clean, resulting in fairly uniform subsidence of the overlying strata, including bed *E*—a factor which contributed materially toward lessening of difficulties incident to subsequent mining of the coal from bed *E*. Timbering and ventilating costs were somewhat higher than they would have been if the underlying bed *D* coal had not been previously exhausted. However, such additional costs were materially offset by a reduction in pumping costs caused by percolation of water from bed *E* workings through the fractured strata, boreholes and shafts to bed *D* workings, from which it was conducted to the surface by way of a drainage tunnel.

(c) In that portion of the property where operations in beds *D* and *E* were conducted simultaneously by different operators with no attempt at planning and coordination, intolerable conditions were created in both mines.

Caving of robbed-out areas in bed *D* mine under active workings in bed *E* mine resulted in subsidences which closed passageways in the latter mine and otherwise rendered unmineable large areas of bed *E* coal.

Likewise, impact and shifting of earth pressures resulting from caving of robbed-out areas in bed *E* mine over active workings in bed *D* mine resulted in fractures, displacement, and squeezing of bed *D* coal and adjacent strata to such extent as to seriously damage workings and to threaten their future usefulness.

Here again litigation ensued, followed by the issuance of court injunctions and counter-injunctions.

Differences were finally compromised after both parties agreed to adoption of plans whereby each would confine its mining activities to areas previously agreed upon wherein operations could be conducted with a minimum of further detriment to either party.

Fractures and subsidence of the coal and roof strata in various sections of bed *E* mine, resulting from robbing of the underlying pillars in bed *D* mine, are shown in Figs. 3-6. Fractures and subsidence of the floor strata are present but are not visible in the photographs. No photographs were taken of disturbances which occurred in bed *D* mine as a result of robbing pillars in immediately overlying areas in bed *E* mine.

CASE NO. 9.

1. **Bed *D*, average 42 in.**
2. **Bed *E*, average 40 in.**
3. **Interval between beds, approximately 45 ft.**
4. **Cover over bed *E*, approximately 300 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) Originally, the mines were owned and operated by different operators and no coordination existed between plans of mining or operations in the two mines.

(b) Bed *D* mine was opened and most of the property was worked over before development of bed *E* coal in that area was commenced.

As bed *E* mine was being developed, pillars were being robbed in bed *D* mine directly under such development work, resulting in an intolerable situation.

The operator of bed *E* mine purchased the underlying bed *D* mine and planned not to rob any more pillars in the latter mine until after the immediately overlying pillars in bed *E* mine were robbed.

The effect of robbing pillars and the resultant caves in bed *E* mine was the creation of such pressure and crushing action on bed *D* pillars and adjacent strata so that large areas had to be abandoned and the coal permanently lost.

CASE NO. 10.

1. **Bed *B* mine, average 30 in.**
2. **Bed *C'*, average 50 in.**
3. **Interval between beds, approximately 110 ft.**
4. **Cover over bed *C'*, approximately 200 ft.**
5. **System of mining, room and pillar.**
6. **General observations:**

(a) Bed *C'* was developed and mined to the limits of permissible exhaustion many years before mining operations were undertaken in bed *B*. Both mines were self-draining.

(b) No effects whatever of the prior mining of the coal from bed *C'* were apparent in the workings in bed *B* mine.

These ten typical cases, which were selected to exemplify some of the major problems of economics and recovery involved in mining two or more beds of coal simultaneously or one after the other, may be divided into the following three categories.

Category A. Cases in which the upper of two or more beds of coal in vertically-adjacent areas was mined to or near the limit of permissible exhaustion before mining operations were commenced in an underlying bed within that area.

Observed cases falling within this category showed no evidence of prior exhaustion of the coal from an upper bed and therefore presented no special problems of economics and recovery, except that in cases where the interval between the beds of coal worked ranged from approximately 40 to 50 ft, e.g., *C'* to *D* and *D* to *E*, regions of high pressure were frequently encountered in the workings underlying those in which large and extensive areas of pillars had been left standing.

All observed workings were self-draining but if they had not been, special precautions would have been required to guard against sudden and dangerous inrushes of water from overlying flooded areas. Flooding is a principal, though by no means deterring, adverse factor incident to the conduct of mining operations in this order.

Category B. Cases in which the lower of two or more beds of coal in vertically adjacent areas was mined to or near the limit of permissible exhaustion before mining operations were commenced in an overlying bed within that area. Among problems of recovery and economics are the following:

1—Unavoidable damage to which the upper bed or beds of coal and adjacent strata were exposed as a result of prior removal of all of the coal from a lower bed without provision for adequate support for the overlying strata. Such injury or damage consists of subsidence, fracture, and parting of the overlying strata and beds of coal within varying ranges.

2—Avoidable damage to an overlying bed or beds of coal and adjacent strata caused by large and extensive pillars of coal left standing in otherwise worked-out areas in an underlying bed.

Injurious effects observed in overlying mine workings were humps in the floor, necessitating extra grading; concentrations of weight of the overlying strata, which caused crushing of the coal and squeezes; also undue fractures in the coal and its adjacent strata, which increased timbering and ventilating costs; and loss of coal beyond normal, unavoidable losses.

However, where pillar extraction in a lower bed of coal had been reasonably complete and ample time allowed for subsidence of the overlying strata, there were large areas in the workings in an upper bed of coal where subsidence of that bed and its adjacent strata was so uniform that there was little evidence of previous mining out of the lower bed. However, lines of successive caves in the lower workings were frequently found as troublesome fracture lines in an upper bed of coal and its adjacent strata.

Escape of gases from the fractured coal and adjacent strata is an ever present hazard which requires constant and close supervision.

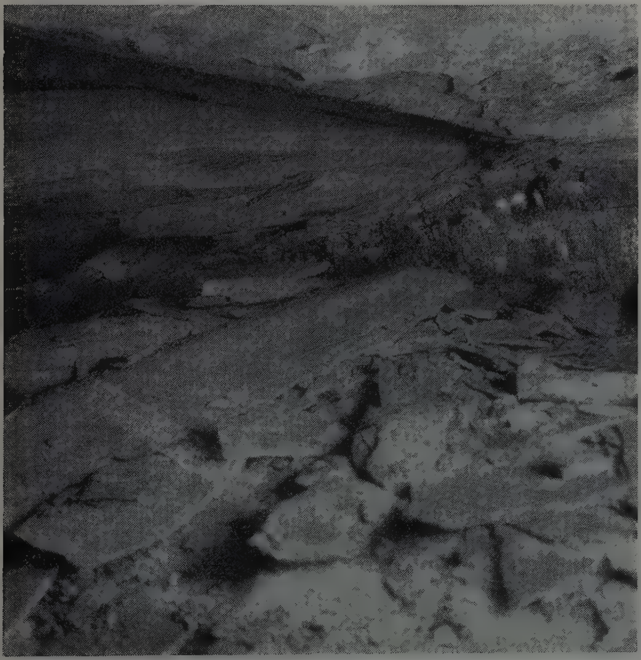

Fig. 3—Fractures and subsidence of the coal and roof strata in bed E mine resulting from robbing of the underlying pillars in bed D mine. Interval approximately 45 ft.

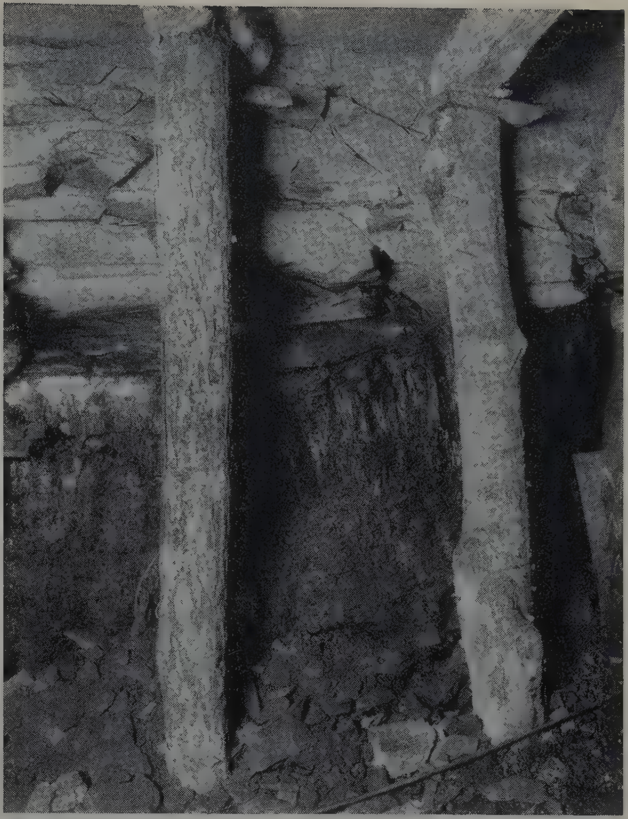

Fig. 4—Fractures and subsidence of the coal and roof strata in bed E.

Offsetting, in part, these adverse factors, drainage problems in cases falling within this category generally were found to be simplified because the water generated in the workings in the upper bed of coal percolated through the fractured strata into the lower worked-out areas or was conducted there by means of boreholes or shafts.

Category C. Cases in which attempts were made to mine two or more beds of coal simultaneously in vertically adjacent areas.

The record of observed cases within this category, particularly those in which the interval between the beds of coal mined was approximately 165 ft or less, is generally one of failures. These failures ranged from local, though quite costly, displacement and/or crushing of the coal and adjacent strata in the workings in one bed of coal or the other, or both, to cases in which large areas of coal and mine workings were so seriously disturbed that permanent abandonment was required.

Notable among observations made in connection with cases falling within this category are the following:

1—Intervals of as much as approximately 200 ft, e.g., those which prevail between beds *B* and *E*, do not necessarily provide protection for mine workings in the upper bed of coal if the supporting pillars are completely extracted from the workings in the lower bed. (An example of an apparent exception is noted in Case No. 5, possible reasons for which were the thinness of the lower bed of coal, the relatively slow rate of exhaustion (hand-loading under adverse conditions), and the frequent occurrence of "bottom rolls" over which the coal was frequently not mined. All of these factors resulted in the reduction of the height of caves in otherwise robbed-out areas.)

2—Complete extraction of the pillars from the workings in a lower bed of coal is practically certain to result in serious damage to, or possible destruction of, immediately overlying workings in any bed of coal within a vertical range of at least 165 ft, which is the approximate average interval between beds *B* and *D*.

3—Complete extraction of the pillars from the workings in an upper bed of coal is almost certain to result in serious damage to, or possible destruction of, immediately underlying workings in any bed of coal within a vertical range of at least 45 ft, which is the approximate average interval between beds *C′* and *D* and beds *D* and *E*. It is probable that similar effects would have been produced in cases involving beds of coal separated by somewhat greater intervals, although supporting evidence to that effect is not indicated by these cases, nor is it material in view of the possible destructive effects referred to in Observations Nos. 1 and 2.)

The three foregoing observations refer to effects produced in coexisting "mine workings" in two or more beds of coal within vertically adjacent areas. They do not apply to virgin beds of coal within such areas that might be similarly exposed to the effects of mining in another bed or beds of coal, that being a subject referred to under the headings Categories A and B.

Suggestions

Experience gained from the cases cited, and many others which have been observed, suggests adherence to the following general principles in cases, as defined, where the mining of two or more beds of coal on the same mining property, either simultaneously or consecutively, is contemplated or in progress.

The simultaneous mining of two or more beds of coal in vertically adjacent areas of a given mining property should be avoided generally in cases where the interval between the beds being mined is approximately 160 to 180 ft or less (except necessary development work). In cases where such interval exceeds approximately 160 to 180 ft, a study of local conditions is warranted before simultaneous mining in vertically adjacent areas is undertaken.

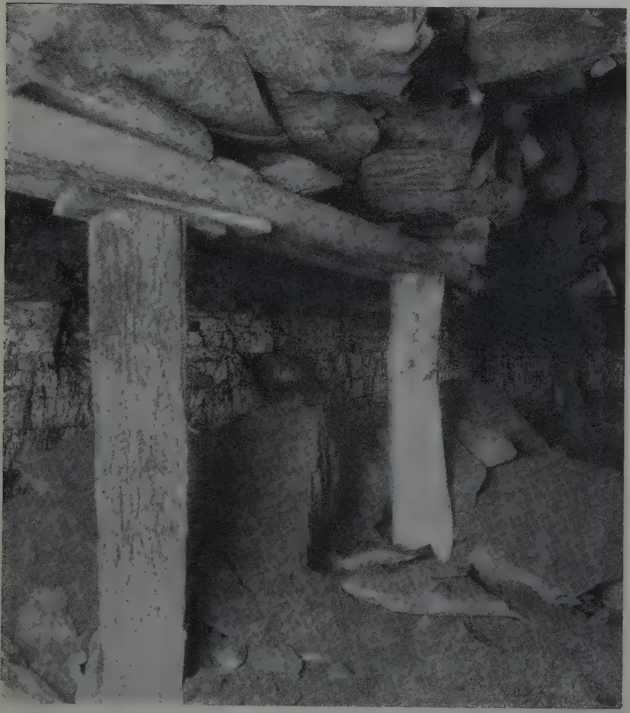

Fig. 5—Fractures and subsidence of the coal and roof strata in bed E.

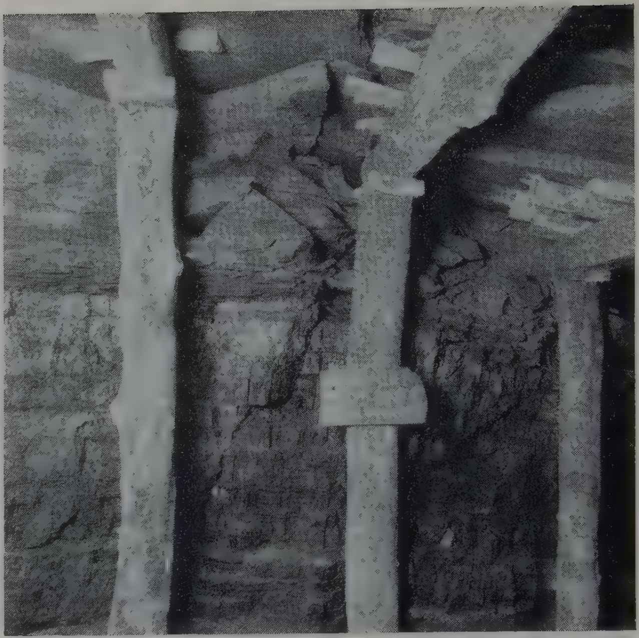

Fig. 6—Fractures and subsidence of the coal and roof strata in bed E.

In cases where it is found desirable to mine two or more beds of coal simultaneously on the same mining property, but not in vertically adjacent areas, the following general plan of procedure is suggested.

Subdivide the property into areas where mining operations (except necessary main entry development in another bed or beds of coal) will be confined to one bed at a time, preferably the uppermost bed first. Continue mining operations in that bed and area to the limit of permissible exhaustion. After subsidence has been completed sufficiently, other workable beds within that area may be mined in sequence while operations in the (preferably) uppermost bed advance into and progress in another area.

The cycle of simultaneous operations thus started may be continued until all workable beds of coal on the property have been exhausted.

Columnize all main entry pillars in each bed of coal worked.

When necessary, provide barrier pillars along such entries in excess of pillars normally required in cases where only one bed of coal is to be mined.

Allow for the effects of draw in the planning and execution of all work.

Require clean extraction of the pillars, which will result in more nearly uniform subsidence of the overlying strata of coal and rock and distribution of earth pressures, as an aid to subsequent recovery of the coal from closely overlying and/or underlying (approximately 50 ft or less) workable beds.

Plan, coordinate and execute all work carefully that interference of mining operations one with the other will be minimized.

Fullers Earth, A General Review

by R. C. Amero

FULLERS earth is a general name applied to claylike minerals that have high natural adsorptive powers. They are usually distinguished from ordinary clays by a higher content of combined moisture and a lower apparent density. The definition is based on the ancient use of the material for fulling, or cleansing woolen cloth of oil and grease; a more modern definition should mention the ability to decolorize oil and should differentiate between fullers earth, which is naturally active, and certain bentonites which only develop decolorizing power after being leached with strong mineral acids. The distinction between fullers earth and activable bentonite is not a sharp one, since all types of intermediate clays can be found that are more or less naturally active, yet more or less responsive to acid treatment. The differences seem to depend somewhat on the amount of leaching that has been effected by natural ground waters.

Physical Properties

The principal mineral constituent of activable bentonite is montmorillonite, a hydrous aluminum silicate to which the following formula is generally assigned:

$$Al_2\,Si_4\,O_{10}\,(OH)_2 \cdot 12H_2O$$

Fullers earth, as well as bentonite, is usually composed of some variety of montmorillonite, but in the United States the largest tonnage of fullers earth is mined from deposits of attapulgite. Attapulgite is a clay mineral composed of monoclinic crystals of colloidal size, having a fibrous shape similar to that of asbestos. A sectional model of the attapulgite "molecule" is shown in Fig. 1. A single unit is about 18.0 Å wide and 12.9 Å high. The length of the molecule is variable, but this same molecular pattern is repeated to form needlework crystals, rather than flakes, as is characteristic of the montmorillonite crystals. Attapulgite has been assigned the following formula,[1] although actually, the magnesia has been largely replaced by alumina:

$$(OH_2) \cdot (OH)_2 \cdot Mg_5 \cdot Si_8 \cdot O_{22} \cdot 4H_2O.$$

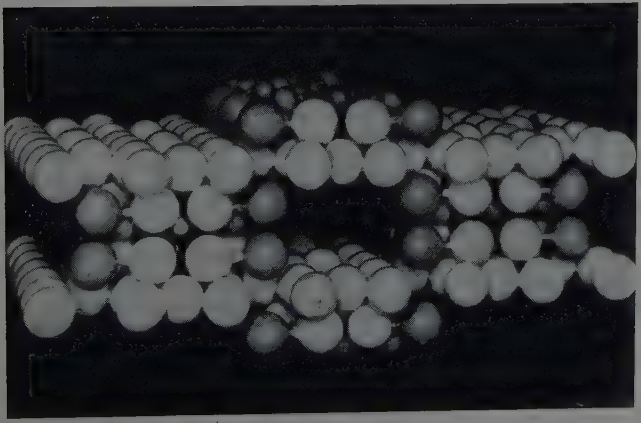

Fig. 1—Model of attapulgite structure.

The fibrous nature of the mineral is clearly shown in the electron micrograph, Fig. 2. Although the deposits contain 5 to 30 pct quartz and carbonate minerals, impurities in the commercial products are held at a low level, principally by selective mining on the basis of oil decolorizing efficiency.

The particle size distribution of this fullers earth has been estimated from settling rates, using sodium citrate as a dispersing agent.[1] Since Stokes' law only applies to spherical particles, the equivalent particle diameter calculated below is probably more nearly an index of diameter rather than length of the needlelike crystals:

Equivalent Particle Diameter	Distribution, Pct	Arbitrary Size Classification
0.05 mm	0.80	0.80 pct sand
0.05-0.02 mm	10.96	42.10 pct silt
0.02-0.01 mm	9.65	
0.01-0.005 mm	11.40	
0.005-0.002 mm	10.09	
2-0.5 micron	10.67	56.58 pct clay
0.5-0.2 micron	29.09	
0.2-0.05 micron	14.86	
0.05 micron	1.96	

The differential thermal analysis shown in Fig. 3 is unique and supports the theory that this is a distinct mineral species.[2, 3] Other general properties of the mineral as shown in Table I.

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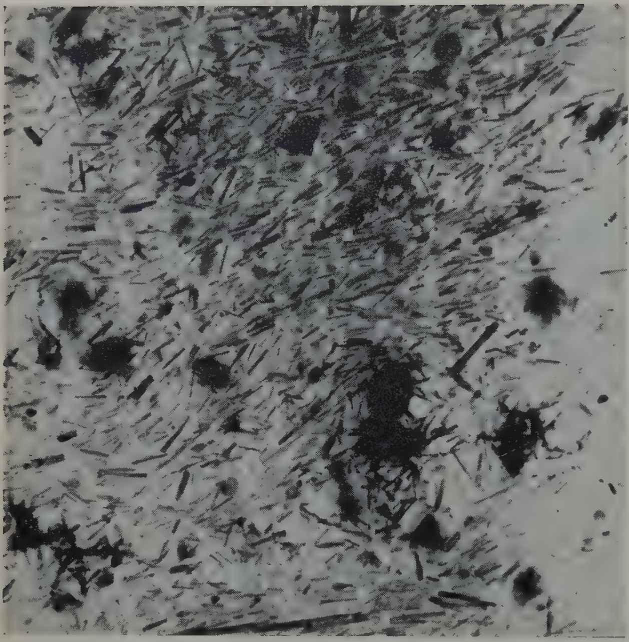

Fig. 2—Micrograph of attapulgite.

To control the properties of fullers earth as an article of commerce, producers depend on three major processing treatments; extrusion, drying, and milling. In the extrusion process, the wet plastic clay is extruded in the shape of rods about ¾ in. diam before drying and milling. The crushed crude clay is kneaded with water in a pug mill, then fed through an auger-type extruder, similar to those employed to form bricks, except designed to operate at higher pressures. By this treatment, laminations in the clay are disrupted and the particles are twisted into a more random orientation. This lowers the apparent density 3 to 4 pct and increases the porosity and the oil decolorizing capacity by about 10 to 25 pct.[4]

The importance of controlled drying is shown in Fig. 4. For a colloidal product that will disperse readily in water, it is necessary to keep drying temperatures below 250°F. To develop maximum adsorptive capacity for moisture vapor, a drying temperature of 600°F seems best, whereas for oil decolorization by the percolation process, an activation temperature of 1000° to 1200°F is necessary.

Table I. General Physical Properties Florida-Georgia Fullers Earth[1]

Refractive Index: Alpha—1.506 to 1.522
Gamma—1.530 to 1.540
Birefringence: 0.021 to 0.032
Optical Character: Negative
Hardness: Approx. 2.0
Base Exchange Capacity: 25-30 milliequivalents per 100 g
pH: 7.9 to 8.2 in distilled water
Specific Gravity: 2.2 to 2.4

Typical Chemical Analysis, Pct:	
SiO_2 (as silicates)	2.36
SiO_2 (as silica)	10.11
SiO_2 (hydrated)	2.42
SiO_2 (combined with bases)	42.63
Al_2O_3	15.43
Fe_2O_3	4.95
FeO	0.30
CaO	1.75
MgO	2.44
Na_2O	0.27
K_2O	0.66
CO_2	0.84
Loss below 105°C[a]	6.39
Loss above 105°C[a]	9.45
	100.00

[a] Moisture content depends on the degree of calcination; this analysis is based on an average commercial product.

Absorbent grades used as floorsweeping compound will "mud up" in water unless heated above 700°F.

For many years, fullers earth was processed primarily for oil decolorization, and the hydrous colloidal form was practically unknown. As a plastic clay or an electronegative colloid, it compares favorably with the swelling bentonites. Bentonite suspensions are more thixotropic and more effective for sealing off a porous surface, but the fullers earth suspensions are more viscous and more resistant to the coagulating effect of electrolytes. Colloidal fullers earth can be expected to have many industrial uses as its properties become more widely known.

Milling reduces the fullers earth to granules or powders in a series of crushing, grinding and sifting operations. Coarse sizes and closely graded fractions offer less resistance to fluid flow. Smaller sizes are less costly and usually have higher adsorptive capacity since they afford more intimate contact between the fullers earth and the fluid being treated. Grades from 2/4 to 16/30-mesh are used if gases or vapors are to be passed through a bed of fullers earth. Sizes from 8/16 to 80/100-mesh are preferred if liquids are to be treated in the same manner. Powdered grades, ranging down to 95 pct finer than 325-mesh, are used if the earth is to be slurried in a liquid.

Floorsweeping compounds are of wide mesh range for economy, although particles finer than 60-mesh are avoided, since they either cause caking or dustiness. As a filler material for intimate blending, as in an insecticide mixture or plastic molding compound, finely powdered grades are used.

Fullers Earth Deposits

Since all clays are colloidal and most of them will decolorize oils to some extent, it is not surprising that deposits classified as fullers earth are found in many parts of the world. Japan and Great Britain are among the most important foreign producers, but the production and consumption in the United States exceeds that of any other nation. At least 18 states have reported production at some time in the past 60 years; the more important deposits are shown in Fig. 5. All of these are reported to be sedimentary in nature except those of Arkansas, which occur in basaltic dykes. The relative production of the various states in recent years is shown in Table II.

Half of the domestic production in recent years has originated in the Georgia-Florida area, principally from a deposit that forms a practically continuous bed under Gadsden County, Fla., and extends into several adjoining counties in both states. Except where badly eroded, it consists of two layers about 4 ft thick, separated by 2 ft or less of harder, more brittle clay, locally called sandrock. The fullers earth is practically level, and overburden varies with the contours of the land. Economical mining is usually limited to 30 to 50 ft of overburden, although up to 75 ft of overburden has been removed on some occasions. Related deposits of poorer quality extend as far south as Manatee County in Fla., but the deposits farther north in Georgia more nearly resemble the Porters Creek clay of the midwest, which is not an attapulgite.

The attapulgite, or "floridin" deposit is classified as part of the Alum Bluff group in the Hawthorn formation, overlying the Chattahoochee limestone. This deposit, indicated by shaded area in Fig. 5, has never been specifically outlined, but the outcroppings and former commercial operations in Leon, Jefferson, Alachua, Marion, Manatee, and other counties indicate the general shape of the deposit. Although some authorities consider all bleaching clays to have been formed by the leaching of volcanic ash, no volcanic detritus has ever been found in this deposit, and it is probably an erosion product from crystalline rocks of the Piedmont plateau in northwestern Georgia. It is assumed that particles of submicroscopic size were transported by water and deposited on the old Florida peninsula when the sediments of the coastal plain were being accumulated. Continental and marine deposits occur both above and below the fullers earth, indicating that the beds were formed during a transitional period, probably in shallow fresh water along the seacoast.

About 35 pct of domestic fullers earth comes from Texas, and most of this occurs in a deposit that runs practically parallel to the Gulf Coast line, and extends from Grimes County, southeast of San Antonio, through Gonzales to Walker County and beyond. A smaller amount of fullers earth is found in the Taylor marl and Navarro limestone, which are also roughly parallel to the Gulf coast, but are farther inland. The useful portions of these deposits vary from 6 to 20 ft in thickness. Although the Texas fullers earth contains adequate evidence of volcanic origin and montmorillonite structure, it seems to be associated with an ancient coast line,

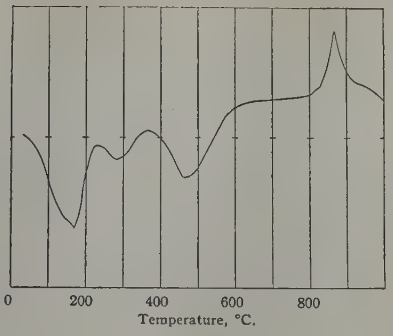

Fig. 3 — Differential thermal analysis of attapulgite.

Table II. Relative Production of States[a]

Year	Alabama	California	Florida and Georgia	Nevada	Texas	Illinois	Other States	Total
1932								
Short Tons	32	100	144,922	—	36,381	—	46,874	228,309
Value, $	288	2250	1,462,794	—	365,374	—	397,021	2,227,727
1933								
Short Tons	266	—	153,703	5,974	31,893	—	32,316	224,152
Value, $	2028	—	1,426,979	61,571	308,096	—	281,966	2,080,640
1934								
Short Tons	—	—	148,319	—	32,763	—	39,182	220,264
Value, $	—	—	1,407,380	—	325,397	—	352,304	2,085,081
1935								
Short Tons	—	—	145,236	—	40,925	—	41,584	227,745
Value, $	—	—	1,491,764	—	391,641	—	346,824	2,230,229
1936								
Short Tons	—	—	139,376	—	46,855	—	44,583	230,814
Value, $	—	—	1,426,346	—	462,656	—	375,976	2,264,978
1937								
Short Tons	—	—	131,100	4,485	49,500	—	41,080	226,165
Value, $	—	—	1,441,588	51,718	473,408	—	329,380	2,296,094
1938								
Short Tons	—	—	91,031	5,984	37,998	—	35,839	170,852
Value, $	—	—	987,391	57,490	358,980	—	303,999	1,707,869
1939								
Short Tons	—	—	91,947	—	38,338	—	36,785	167,070
Value, $	—	—	1,035,066	—	359,058	—	297,731	1,691,855
1940								
Short Tons	—	—	79,898	—	34,039	—	32,631	146,568
Value, $	—	—	917,365	—	277,229	—	276,489	1,471,083
1941								
Short Tons	—	—	91,925	—	77,033	26,676	11,812	207,446
Value, $	—	—	1,075,318	—	713,085	209,577	113,694	2,111,674
1942								
Short Tons	—	—	83,007	—	85,012	30,421	5,804	204,244
Value, $	—	—	1,091,062	—	712,303	264,611	71,694	2,139,670
1943								
Short Tons	—	—	105,647	—	94,137	63,909	7,974	271,667
Value, $	—	—	1,444,936	—	728,141	578,805	118,926	2,867,808
1944								
Short Tons	—	—	128,654	—	111,212	42,277	12,594	294,737
Value, $	—	—	1,833,682	—	916,159	390,346	156,877	3,297,064
1945								
Short Tons	—	—	134,401	—	103,076	43,644	15,227	296,368
Value, $	—	—	1,939,035	—	931,878	403,084	189,916	3,463,913
1946								
Short Tons	—	—	144,212	—	110,693	33,134	10,711	298,752
Value, $	—	—	2,100,652	—	1,157,892	296,637	147,812	3,702,993
1947								
Short Tons	—	—	168,557	—	102,901	37,740	19,870	329,068
Value, $	—	—	2,699,660	—	1,199,726	388,955	372,273	4,660,614
1948								
Short Tons	—	—	188,014	—	92,310	37,942	23,815	342,081
Value, $	—	—	3,224,169	—	1,162,336	410,678	476,668	5,273,851

[a] Tabulated from *Minerals Yearbooks*, U. S. Bur. of Mines.

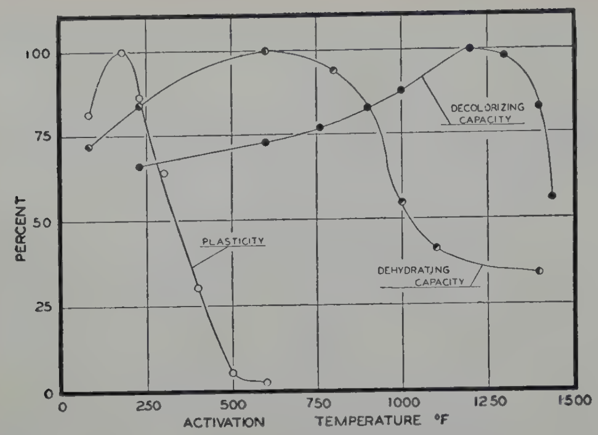

Fig. 4—Effect of drying temperature on properties of fullers earth.

as was the attapulgite described above and the Porters Creek clay described below.

Porters Creek clay runs from Missouri through Illinois, Kentucky, and Tennessee and into Mississippi and Alabama. It is possible that the fullers earth of central and northern Georgia is closely related to this material, although maps of the Porters Creek clay are never extended to include the Georgia deposits. About 10 pct of domestic fullers earth production is mined from the Porters Creek deposit in Pulaski County, Illinois, where it reaches a thickness of 60 ft. Production is also occasionally reported from the other states in which this clay is found. The geological relationship of Porters Creek clay and other domestic fullers earth deposits[5] is as follows:

Cenozoic

- Miocene: Hawthorn Formation, Georgia and Florida, fullers earth.
- Oligocene: Vicksburg Group: Mississippi bentonite; Bentonite clays in Florida, Georgia, Alabama.
- Eocene: Jackson Formation: Georgia bleaching clay, Mississippi bentonite and bentonitic clay; Alabama bentonitic clay (Probably also Texas fullers earth).
- Lisbon Formation: Alabama bentonitic clays (Clarke and Choctaw counties).
- Porters Creek Clay: Southern Illinois, Southeastern Missouri, Western Kentucky and Tennessee, Mississippi and Alabama.

Mesozoic

- Upper Cretaceous:
  - Eutaw Formation: Northeastern Mississippi (Probably also bentonitic fullers earth in Taylor marl and Navarro formation of Texas).

Paleozoic

- Ordovician
  - Chickamauga limestone: Northwestern Georgia, Northern Alabama, Kentucky and elsewhere.

Porters Creek clay and all of the western fullers earths are 50 to 80 pct as efficient as Georgia-Florida clay for decolorizing oil. The exceptions to this rule are limited to certain minor uses, particularly in the refining of vegetable oils. For this reason, the Georgia-Florida material serves the petroleum industry in the entire eastern half of the country, and the lower grade materials are justified principally by lower mining costs and lower freight rates into adjacent refining areas. Fullers earth quality is always established by laboratory scale oil decolorizing tests; no physical or chemical test has been found that will predict or fully explain this decolorizing action.

California and Utah have consistently produced a small amount of fullers earth in recent years, but production from other states has been sporadic, and often for rather specialized purposes. For example, the fullers earth of Massachusetts has been used as a binder for abrasive grinding wheels, and Porters Creek clay from Mississippi was sold exclusively as a floorsweeping compound.

History of the Industry

The use of fullers earth for "fulling" dates back to the Middle Ages in England, and to Greece and Rome in ancient history. Domestic interest in fullers earth has been traced back to 1880, when N. K. Fairbanks and Co., Chicago, learned that clay was being used in the Orient to improve the color of olive oil. After extensive testing with all of the clays available to them at that time, they settled on English fullers earth as the best decolorizing agent for cottonseed oil.

The first discoveries of fullers earth in the United States were reported in 1891 at Alexander, Ark.,

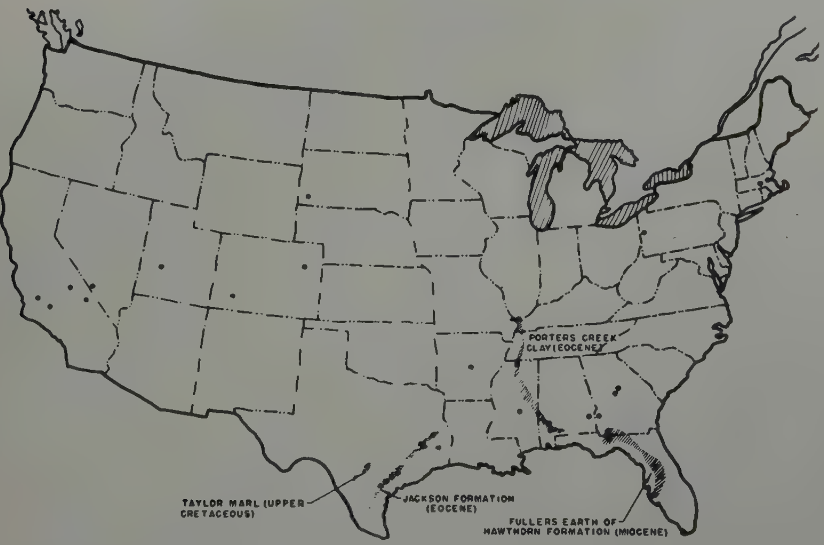
PORTERS CREEK CLAY (EOCENE)
TAYLOR MARL (UPPER CRETACEOUS)
JACKSON FORMATION (EOCENE)
FULLERS EARTH OF HAWTHORN FORMATION (MIOCENE)

Fig. 5—Principal deposits of fullers earth.

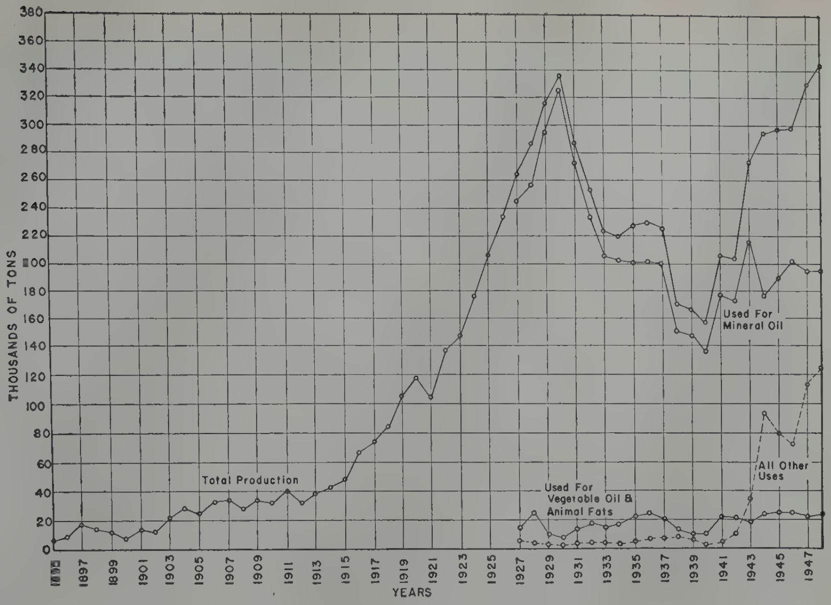

Fig. 6 — Domestic fullers earth production.

Tabulated from *Minerals Yearbook.*

and in 1893 at Quincy, Fla. From an output of 6900 tons in 1895, the production of fullers earth grew with the petroleum industry up to a peak of 335,000 tons in 1930. As shown in Fig. 6, almost 95 pct of this went to petroleum refineries and almost all of the remainder to vegetable oil refineries. Less than 1 pct went to miscellaneous consumers. Whereas oil refining activity stayed at a practically constant level from 1930 to 1935 and then began to expand rapidly, fullers earth consumption dropped steadily to a low of 146,000 tons in 1940. Better regeneration equipment in the refineries lengthened the useful life of the clay; solvent refining and other new techniques reduced the quantity of clay required. Serious competition was being encountered from the more active acid-treated clays and synthetic magnesium silicate, and from the more rugged activated bauxite. A better fullers earth was needed to hold the refinery markets as well as development of new outlets to replace the refining business that was permanently lost. In the interest of self-preservation, some of the larger companies ventured into the production of active bauxite and acid-treated bentonite and started investigations of synthetic adsorbents, particularly magnesium silicates.

The extrusion process mentioned previously was developed about 1936. This was most timely to prevent a widespread changeover to bauxite in the lube oil refineries, since the cost analyses and advantages of each of the two adsorbents were quite evenly matched until extrusion appeared.[6, 7] Fig. 7, which is based on plant records, shows that the less efficient crude fullers earth is more responsive to this treatment than the better crudes. Therefore, it was possible to use many deposits that were formerly too low in quality, and the quality of shipments to oil refineries was not only improved, but also became more uniform. The impact of the extrusion process on the fullers earth industry is demonstrated in Fig. 8, which is a tabulation of the quality of shipments averaged from daily quality control records. The improvement between 1935 and 1937 reflects the adoption of the extrusion process. Bauxite has found a place in the filtration of waxes and certain types of oils, but because of the improvement in fullers earth and the virtual exhaustion of the better deposits of bauxite, it no longer threatens the main uses of fullers earth.

Between 1935 and 1940, new markets were developed for fullers earth as a viscosity-building agent in oil well drilling fluids and as a decolorizing filter aid for used dry cleaning solvents.

From 1940 to 1945, wartime conditions accelerated the demand for all grades in all uses, and at the time when the industry was least prepared to handle it, a remarkable new outlet developed. In 1942, granular fullers earth became popular as a floorsweeping compound for oily, slippery floors. The material did not have to be screened to narrow mesh sizes; quality variations of 10 pct or more could not even be detected by the customer. The demand was enormous. At the same time, but in a less spectacular fashion, a market gradually unfolded for the powdered grades as absorbents and conditioners for the new, complex insecticides such as DDT and BHC. In the Georgia-Florida area, where dust had been accepted as an unsaleable byproduct ever since the refining market had been lost to acid-treated bentonite in 1930, this new outlet was doubly welcome. In the postwar period, the demand for fullers earth has continued to increase with the general business index until tonnages stand once again at the 1930 level.

Changed Production Methods

In terms of tonnage, the industry may have completed a 20-year cycle, but it is not back where it started. Now only 66 pct of the annual production goes to mineral and vegetable oil refineries, instead of 99 pct. In every aspect of the business, machinery and procedures are more modern and economical.

For prospecting, tractor-mounted, power-driven augers have now come up in efficiency and down in cost to the point where they can replace a prospecting crew with hand augers for taking samples in wooded areas and on soft ground. The newly adopted bulldozer and carryall techniques clear woodland and work strip mines more economically, and at higher overburden ratios than steam shovels, draglines, and dinky trains. A dozen men now mine crude earth at the same rate as 50 or 60 men just

10 years ago. Rotary dryers are still preferred for drying the clay, but the trend is to dryers 6 to 7 ft in diam and 80 to 100 ft long, equipped with automatic controls, in place of the manually controlled dryers 5 to 6 ft in diam and 40 to 60 ft long. Better fuel economy and more uniform products are being obtained as a result, especially where clay temperatures of 800° to 1200°F are required for "calcined" products.

Little change has been made in the milling operation in which the dried lumps of clay are ground and sifted to granular sizes on conventional flour mill machinery, with corrugated rolls and wire and silk screens. Since the granular sizes are most valuable, the art of gradual size reduction and closed-circuit grinding is highly developed, and 65 to 70 pct can be recovered in the 16/30 and 30/60-mesh sizes. Perhaps the major improvement in milling operations in the past 20 years has been the installation of automatic weighing equipment to check screening yields and serve as a guide in adjusting mill rolls. The patented technique of Raymond-milling damp fullers earth before the final drying step also deserves mention.[8] By this method, it is possible to produce powders consisting of rounded particles with a minimum of dust and extreme fines; fullers earth treated in this manner will filter liquids at more than twice the flow rate that can be used on an ordinary fullers earth, with the same pressure drop, although both powders are substantially finer than 300-mesh.

In the shipping department, burlap bags or bulk carloads are used for refinery accounts, but paper bags are preferred for fine mesh products and for shipments to nonrefinery accounts that do not want to be bothered returning empty bags for credit. Shipping departments have been modernized to the extent of using valve bags and some carloading aids, such as portable conveyors, but lift trucks, palletized loads, and airveying systems have not yet been generally adopted.

Changed Marketing Methods

When 95 pct of the fullers earth went to a few refineries, one or two salesmen could canvass the whole market with no assistance from advertising or distributors. Most shipments were carload quantities direct from producer to consumer, although orders for smaller quantities were tolerated. The production control laboratory would run tests to assist customers with technical problems; the same laboratory devoted occasional spare time to investigating new uses for fullers earth.

The industry still competes for the oil refining business as its primary outlet. Technical service and engineering data are the salesman's chief aids in this competition. For the other markets, there is an increasing use of generalized literature, eye-appealing advertisements, jobber and distributor connections. This is an acknowledgment of the new markets—well drilling mud, insecticide blending, dry cleaning and floorsweeping—where customers are more numerous, more widely scattered and individually consume smaller tonnages. Research programs have been developed to serve the existing applications and develop new ones. In spite of the stable or declining market in petroleum refining, fullers earth has a future as a colloid, adsorbent, filler, filter aid, catalyst, and absorbent.

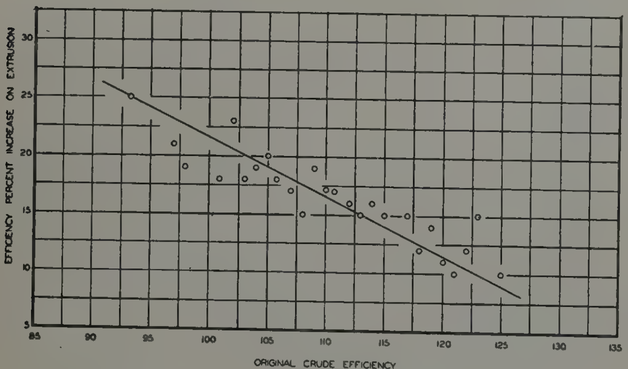

Fig. 7—Clay response to plant extrusion.

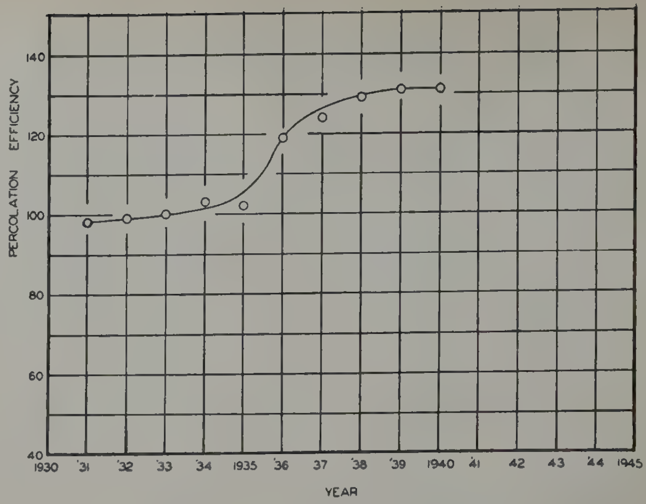

Fig. 8—Average yearly weight efficiency.

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INDUSTRIAL SALTS:

Production at Searles Lake

by J. E. Ryan

TRONA, Calif., is a miniature urban community of some 3500 people, located on the northwest shore of dry Searles Lake in the extreme northwest corner of San Bernardino County, approximately 186 miles north and east of Los Angeles. Since it is situated on the Mojave Desert, a typically desert climate prevails with wide variations in temperature between day and night, extreme daytime summer heat, and cool to cold winters. Rainfall averages somewhat less than 4 in. per year, and dust storms are common. The rate of evaporation, however, is great, amounting to 6 to 9 ft of water per year. The extremely low humidity makes the summer heat of 110°F tolerable with only a mild, temporary discomfort.

Nature of Deposit

During the periods when Searles Basin was flooded, the waters that passed through Indian Wells Valley spread out to form a broad, shallow lake providing, in effect, a settling basin for suspended sediment. The drainage into the deeper and more isolated Searles Basin thus was clarified to a great degree before concentration began. Today, the elevation of the dry surface of Searles Lake is 1618 ft, and the salt deposit measures 5 x 7 miles.

At the eastern and northeastern margins of the main playa zone and just at the foot of the alluvial slope of sand and coarser wash from the Slate Range mountains, a rim of crusted salts rises a few feet above the level of the flat. The deposit is a saline efflorescence composed of salts that were presumably brought up with rising ground waters to be deposited at the surface by solar evaporation. This deposit consists chiefly of trona, and it is after this Trona Reef that the town Trona was named. Strip mining operations have been conducted in the past at infrequent intervals for the recovery of crude trona salts.

The main focal point of interest in Searles Lake from a commercial standpoint is the main salt body located almost centrally in the basin. The exposed portion of this porous saline deposit covers approximately 12 sq miles and averages 71 ft deep. Its interstitial voids, which constitute 50 pct of the total volume, are permeated with a brine, which is in equilibrium with the soluble salt deposits. The brine is the raw material for the operations of the American Potash and Chemical Corp. plant at Trona, shown in Fig. 1. The soluble salt deposits are of interest for their potential values in future technologic development. The brine, which is stratified according to slight differences in density, stands usually within 6 in. of the surface of this exposed, firm, salt body. The surface is usually dry and will support the weight of heavy mobile units and drilling equipment. Occasionally, however, surface waters from the higher watersheds encroach upon the main salt body during infrequent periods of precipitation on the surrounding mountains. This water dissolves surface salt, becomes a dilute brine, and has been observed to stand as high as 18 in. above the salt surface when undisturbed. Windstorms will shift the water back and forth across the lake surface.

The exposed salt body is surrounded by additional submerged areas of commercial soluble salt deposits covering some 20 sq miles, hidden from view by marginal playa mud. These vary in depth up to as much as 30 ft. Thus, the outline of exposed and submerged salt deposits of commercial value is estimated to cover a total area of 32 sq miles, which is roughly circular but slightly elongated from northwest to southeast. It has been estimated that each square mile contains about 100 million tons[2] of alkali salts.

The results of drill borings in the past 15 years have brought to light the interesting fact that the main salt body lies superimposed on an impervious mud deposit from 10 to 15 ft thick containing relatively little soluble salt. Under this deposit lies a second soluble salt body 35 ft deep. The lower salt body is interspersed with numerous insoluble mud lenses and its composition is considerably different from that of the primary, or main salt deposit. Recent drill borings have not penetrated beyond 300 ft. They have, however, revealed that underlying the lower salt body, the mud sediments carry deposited minerals of trona, nahcolite, mirabilite and much less soluble carbonates or sulphates of calcium and/or magnesium. This structure is shown in Fig. 2.

Current Lake Survey Program

Several hundred holes have been drilled in the deposit. However, to carry out a thorough and carefully correlated study of the composition of the soluble salts and other minerals in the dry lake basin, a drilling program was inaugurated recently and is now nearing completion. In this survey, pattern drill holes are sunk at regular ½-mile intervals to a depth of approximately 150 ft.

Drilling equipment consists of a No. 51 C. P.

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Geologic History

Searles Lake occupies a basin lying between the Argus Range on the west or northwest and the Slate Range on the east. The basin was occupied by at least one deep lake, traces of which are evidenced by the records of ancient shore lines and water channels that stand forth on the surrounding ranges.[1] While the waters stood at their highest position, the basin was flooded to a depth of 635 ft above the present dry lake surface, and covered an area of 384 sq miles. The lake historically extended into Salt Wells and Indian Wells Valleys. For several thousand years Searles Lake must have been one of a series, or chain of lakes joined by rivers, which resulted from the overflow of Owens Lake to the northwest. The flood waters passed from Owens Valley, at an elevation of 3760 ft, through Indian Wells Valley into Searles Valley at an elevation of 2260 ft. The waters overflowed Searles Valley into Panamint Valley and finally into Death Valley. The fact that Searles Basin for a time overflowed into Panamint Valley and beyond evidently did not prevent the accumulation and deposition in the Searles Basin of an immense mass of solid crystalline salts and mother liquor brine.

The main salt deposit of Searles Lake represents a crystalline product of desiccating solutions, of which the desiccation has not yet been carried to the final stages. Geological survey indicates that this last period of desiccation, since Owens Lake last ceased to overflow, was somewhat less than 4000 years ago. The residual mother liquor of the Searles deposit still constitutes a large proportion of the mass. It does not lie on the surface of the salts but rather permeates the deposit that has been built up in a loose or open-texture manner. Evaporation at the surface is counterbalanced by percolation or seepage from bordering slopes.

Mitchell Diamond Size "A" drill rig, and a Sullivan Size "A" core drill. Both rigs are equipped with positive screw feed, chucks, and quills for standard drill rods. The rigs are mounted on wide-tired standard wheels for mobility and are designed for surface drilling.

Core barrels ranging in sizes from 2-in. to 15-in. OD may be used as desired. Since large crystals, frequently encountered, may wedge in a 2-in. core barrel, the larger core barrels generally are used. In the present field survey the 9-in. core barrel, shown in Fig. 3, has been made standard equipment. Many types of core barrels have been tested, but the most suitable is the simplest in design. This core barrel is a piece of thin-walled seamless tubing, 4.5 ft long. The cutters consist of tempered steel saw teeth cut in the end of the tubing, with points of 3/16-in. Stoodite, a hard facing alloy, welded to the cutting edges. Circulation for removing cuttings and cooling the bit as used in standard core drilling methods is not required. The bit is cooled by the brine that permeates the salt body.

In the top of the core barrel, a vent, equipped with a plug, is provided through which air may be introduced under pressure to displace core contents. This plug is inserted in the vent after the barrel has been lowered below the brine level, 6 to 8 in. below the exposed salt surface. As drilling progresses, the core enters the barrel, displacing the brine from within. The displaced brine is forced down along the wall of the barrel and out, carrying the cuttings along with it. Cuttings that remain behind pack tightly around the core and serve to hold it in place with no need for a core-catching device since the core barrel is brought to the surface at the end of the drill run. The core is then freed from the barrel by opening the vent and applying pressure if necessary.

Core recovered is logged and sampled for chemical analysis. The core is also subjected to field tests for porosity. Calculations based upon the results of chemical analyses of the cores are also used to determine porosity. Both sets of data are in good agreement.

For cleaning holes following drilling operations, the air-jetting technique is employed. A portable air compressor capable of delivering at 125 psi is employed to bring up about 400 gal per min of brine. In certain areas, it is necessary to cement casing in the hole to depths up to 30 ft to prevent caving of surface material.

After a hole is drilled, the brine is allowed to stand undisturbed for several days. Samples of the brine are then taken at regular depth intervals to determine its quality at respective depths. Fluorescent dye, color tests tend to indicate the existence or absence of brine flows toward low pressure areas, and by systematic sampling and testing at regular intervals, changes in brine composition in affected areas due to pumping are determined.

Brine-Producing Wells

To drill a brine-producing well, a 10, 15, or 18-in. diam reamer with Stoodite cutting bits is selected. The 18-in. type reamer is shown in Fig. 3. Cuttings are flushed from the hole by a continuous flow of air. By this procedure, a hole may be drilled to a depth of 80 ft within a 16-hr drilling time. The well is cased with 10-in. pipe to within 10 ft of the mud interface underlying the upper salt body, and the annular space between the casing and salt deposit is filled with concrete. This procedure is followed to preclude draw-down of upper strata or surface brines into the well. The well casing, which is made to stand 1 ft above the lake surface, is built up with standard 10-in. flanged pipe to a height 4 ft above the lake surface.

Individual pumps are mounted directly on the top of the flanged 10-in. standard pipe. The pumps are vertical, turbine type, fitted with individual 15 hp motor drive and weatherproof electrical equipment. The shafting and pump bowls extend down about 10 ft. The pump will deliver about 100 gpm against 140 psi of head, or line pressure. Each well is connected through a check valve to the main header, which carries brine into the plant storage tanks. A total of 34 producing wells, spaced at 500-ft intervals, are pumped continuously to supply 17,000 tons daily of raw brine to the main plant evaporative process. The brine is pumped 2 to 3 miles through 8-in., 10-in., and 12-in. branch iron pipe headers, and finally over a distance of 4 miles

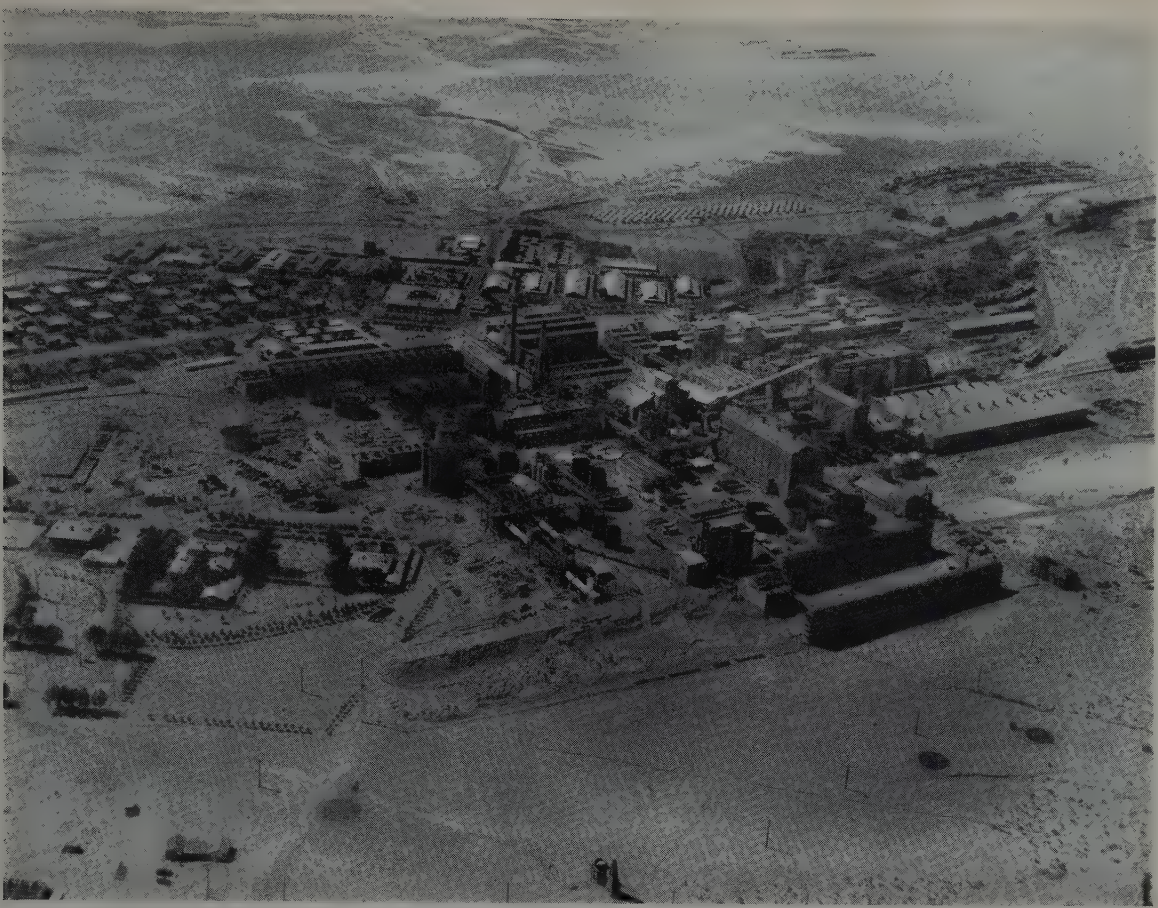

Fig. 1—Aerial view of Trona plant from northwest. Recently constructed carbonation plant is in foreground. Abandoned solar evaporating pond is to right of plant effluent on Searles Lake in background.

through buried 14-in. main transite line to the brine storage tanks.

In addition, eight producing deep-wells are pumped continuously to supply approximately 8000 tons per day of lower structure brine to a carbonation process. These wells are spaced at 500-ft intervals and are cased with stainless steel 10-in. tubing to depths of 115 ft below the lake salt surface. The deep-well brine is pumped through concrete-lined 10-in. iron pipe into a buried 12-in. transite header, which leads to the carbonation plant primary towers.

Commercial Plant

The Trona plant represents an investment of about $32 million, and employs some 1500 workers in the production and marketing of 1800 tons daily of sodium, potassium, and lithium salts, including sodium borates, and also bromine and various bromides. The plant operates two processes that are relatively independent of one another, each of which uses a separate brine from different depths in the lake salt deposits as a source of raw material. Table I shows the composition of upper and lower structure brines as pumped.

The main plant process is a cyclic, closed circuit, in which raw brine from the primary salt deposit is subjected to successive evaporative and cooling steps to effect the segregation of selected salts through fractional crystallization. The carbonation plant process consists of carbonation, utilizing waste flue gas from the power plant boilers as a source of CO_2 to precipitate bicarbonate of soda, which is calcined to soda ash with the recovery of CO_2 gas. This gas is recycled to the secondary carbonating towers and the soda ash is bleached and further processed for the removal of impurities. The carbonation plant process recently was covered in a paper entitled *"The Utilization of Natural Brines from California Dry Lakes for the Manufacture of Soda Ash by Carbonation"*, by F. H. May and M. L. Leonardi, presented before the AIME Los Angeles meeting, October 1949. Therefore, the balance of this paper will be confined to the main plant cyclic process.

Main Plant Cyclic Process

Concentration: In the main plant cycle, raw brine from the storage tanks is mixed with process end liquors to become *evaporator feed* liquor, and, in theory at least, it is ultimately evaporated to dryness as it is recycled some 38 times through

Table I. Composition of Upper and Lower Structure Brines, as Pumped

Constituent	Upper Structure, Pct	Lower Structure, Pct
KCl	5.02	2.94
Na_2CO_3	4.80	6.78
Na_2B_4O	1.63	1.96
Na_2SO_4	6.75	6.56
NaCl	16.06	15.51
Na_2S	0.08	0.38
Li_2O	0.015	0.006
KBr	0.12	0.08
WO_3	0.007	0.004
I_2	0.003	0.002
P_2O_5	0.070	0.044
F	0.002	0.002

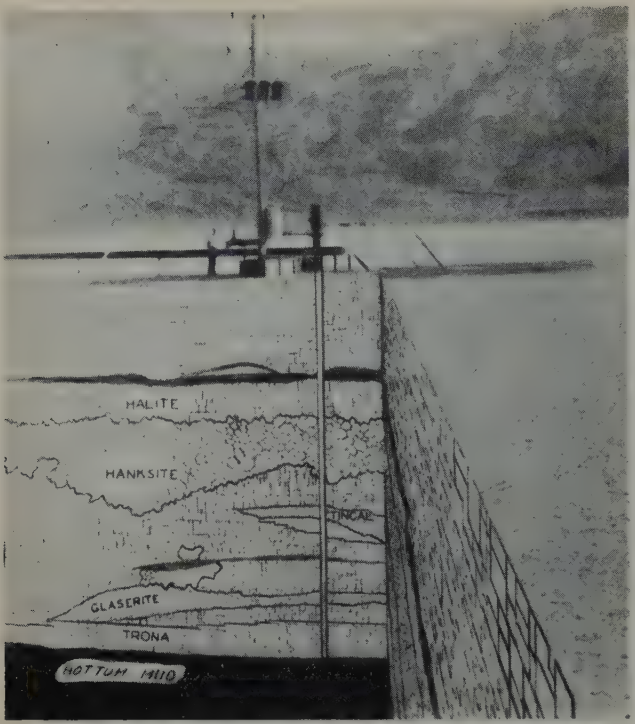

Fig. 2—Cross-section of upper salt deposit showing a typical well casing and pump mounting. The designation "Bottom Mud" is a misnomer and should properly read "Parting Mud" separating the upper 71-ft salt body from the lower 40-ft salt deposit, both of which are permeated with saturated brines.

triple effect counter-current evaporators and crystallizer units. Heat supply to the evaporators is 32-lb exhaust steam from turbo-generators, which are rated at 31,000 kva. The generators are driven by steam pressures ranging from 225 to 450 psi. Steam is generated by Babcock & Wilcox boiler units, which consume 1 bbl of oil per 1000 gal of raw brine pumped.

Evaporation of the brine results in the precipitation of sodium chloride from its saturated solution. In addition, heating and evaporation of the brine causes the precipitation of burkeite, a double sodium salt of carbonate and sulphate which is inversely soluble. As the precipitated salts are removed from the saturated liquor by suitable elutriation equipment, the liquor becomes increasingly concentrated with respect to the directly soluble potassium chloride and sodium tetraborate constituents. This concentrated liquor, thus freed of a good portion of the sodium chloride, carbonate, and sulphate, is allowed to become concentrated fourfold with respect to dissolved potassium chloride and sodium tetraborate values at a temperature of 220°F as it is discharged from the first effect evaporator.

Potash Crystallization: Potash crystallization is essentially a cooling, settling, and filtering operation wherein the concentrated liquor is diluted and then rapidly cooled to 100°F. Rapid cooling takes place in three-stage, vacuum, flash-coolers designed for approximately 20-min retention time.[3] The dilution of the hot liquor compensates for evaporation during vacuum cooling to prevent sodium chloride precipitation. Raw brine is used as the condensing medium because of its lower vapor pressure. Although the point of incipient borate precipitation under these conditions is 135°F, advantage is taken of the fact that borax tends to supersaturate. The muriate of potash crystals, thus precipitated relatively free of borate contamination, are thickened in cone settlers and filtered off in basket-type centrifugals. Approximately 650 tons per day of muriate of potash on an anhydrous basis are produced. It is given a displacement wash and is dried in rotary oil-fired kilns enroute to storage to await shipment as agricultural grade (61 pct K_2O) muriate of potash. A portion of the product is recrystallized to produce chemical grade muriate of potash. From this operation the KCl solution is treated in Kubierschky-type towers to remove bromine. The recovered bromine is refined and marketed as liquid bromine or alkali bromides of USP quality. A portion of the muriate production also is made into potassium sulphate by reaction with crude burkeite or sodium sulphate. Filtrate from the potash centrifugal baskets is combined with the cone overflow liquor and forwarded to the borax crystallizers.

Borax Crystallizers: At the borax plant, end liquor from the potash plant is diluted with a small amount of borax refinery end liquor and is distributed to six vacuum ammonia-cooled crystallizers equipped with Kinney pumps for the removal of N.C. gases, wherein the liquor is cooled to about 80° F.[3] At this lowered temperature, crude borax is precipitated and thickened. The thick sludge is filtered over Oliver filters. Filtrate from the filters is combined with overflow from the thickener and is pumped, via heat exchangers at the potash crystallizers, back to the main plant evaporator feed storage tanks. This completes the main plant cycle.

The crude borax filter cake is brine-leached to remove soluble phosphate impurities and then is

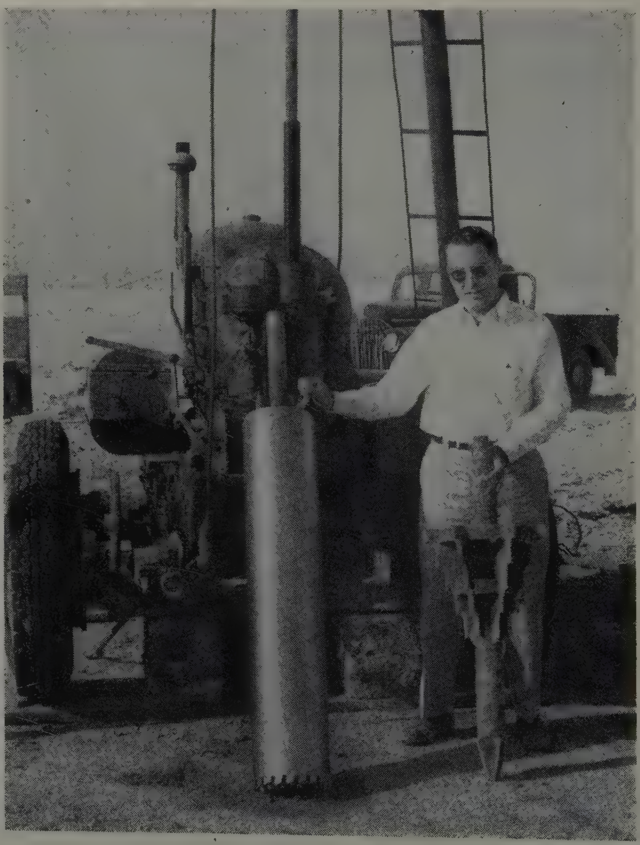

Fig. 3 — 48x9.5-in. core barrel 18-in. reamer, Sullivan model 22-HD core-drill in background.

recrystallized for the manufacture of refined borax. Part of the refined borax is treated with sulphuric acid to produce technical and USP grades of boric acid. In addition, a substantial portion of the refined borax production is dehydrated by heating ultimately to fusion temperatures in the recently expanded pyrobor plant to produce "Pyrobor," a crystalline anhydrous borax. Total borate production is 300 tons per day.

Evaporator Tail Salts Processing: At the main plant evaporators, such as that shown in Fig. 4, the sodium chloride is separated from burkeite by means of suitable elutriators designed to operate in series as hydraulic classifiers. Coarse sodium chloride material discharging from the underflow of this salt trap is washed free of its brine entrainment on vibrating screens and is sluiced back to the lake to the extent of 2500 tons daily. The burkeite fines, or sodium double salts of carbonate and sulphate, are filtered on Oliver filters and given a brine displacement wash with end liquors from the soda products plant. The washed filter cake, which amounts to approximately 1300 tons daily, is repulped in these same end liquors, which serve as a vehicle in transporting this raw material feed to the soda products plant.

Soda Products Process

At the soda products plant, the burkeite double salt of sodium carbonate and sulphate is leached for free sodium carbonate in excess of that combined as burkeite in a cold saturator tank. The slurry is then treated with sodium chloride in a hot saturator tank equipped with steam coils for burkeite precipitation. The burkeite solids are then thickened, filtered, and dissolved in water. Because of its inverse solubility, dissolution of burkeite necessitates additional cooling equipment to operate in closed circuit with the dissolver to remove the evolved heat of solution characteristic of this material.

Salt Cake Production: The dissolved burkeite slurry, which is made up to analyze 30 pct of dissolved salts by means of sensitive pneumatic density-controlled devices, is passed through the lithium plant for the removal of undissolved lithium phosphate suspension. The clarified burkeite liquor is forwarded to glauber salt crystallizers, wherein the liquor is cooled to approximately 78°F. At this point of glauber salt precipitation, crystals are filtered and intermixed with sodium chloride in converter tanks. Anhydrous sodium sulphate or salt cake is thus produced by dissolution of sodium chloride in the water of hydration, which constitutes 56 pct by weight of the glauber salt mass. The reaction is endothermic and the degree of refrigeration so obtained is utilized in the barometric condensing system of the glauber salt crystallizers, thus forming a regenerative refrigeration cycle.

Filtrate from the glauber salt filters is forwarded via surge tanks to vacuum ammonia-cooled crystallizer tanks. Cooled crystallizer discharge is then forwarded to crude sulphate precipitator units, wherein treatment with sodium chloride results in the precipitation of much of the residual sodium sulphate. End liquors from these precipitator units are returned to the main evaporator plant for sluicing burkeite filter cake raw material feed to the soda products system. The crude sodium sulphate is thickened, filtered, leached for chloride impurity, and further thickened. Thickener overflow is discarded as sulphate end liquor, and the underflow is combined with the anhydrous sulphate production from the glauber salt crystallizers. This material is filtered and dried in rotary oil-fired dryers and stored in concrete silos at rates up to 600 tons per day to await shipment.

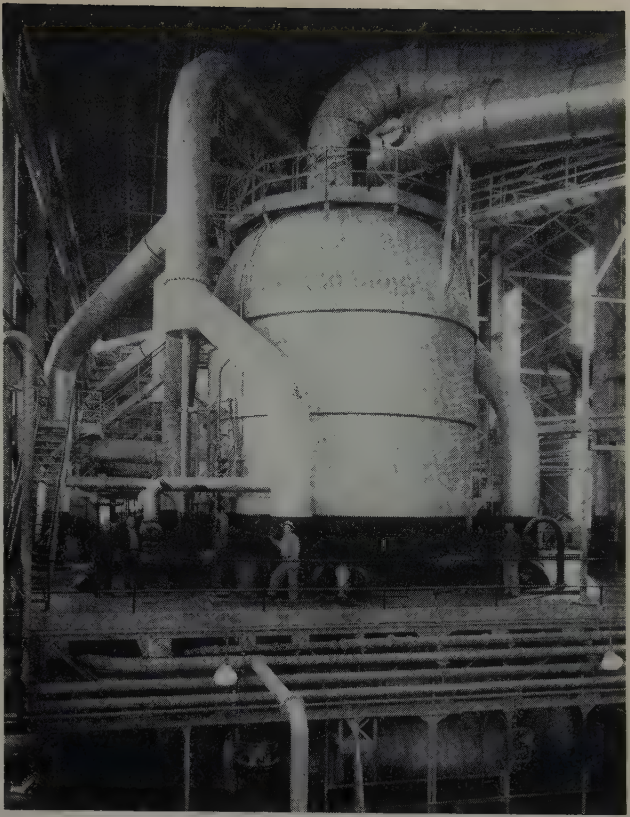

Fig. 4—One of nine evaporator pans, which measure 58 ft high by 26 ft in diam. Note external heater units and vertical, turbine-type circulation pumps in lower foreground.

Soda Ash Production: The 60-ft Dorr thickeners which are fed from the hot and cold saturator tanks and which discharge chloride-free burkeite underflow solids to the sulphate refinery were described. The clarified thickener overflow is combined with filtrate liquor from the burkeite filters to become raw material feed to the carbonate refinery of the soda products plant. This hot carbonate liquor is first cooled in two-stage barometric vacuum crystallizers for sodium chloride precipitation. The sodium chloride is thickened and routed where needed to the sulphate precipitators and/or hot saturator tank. The clarified liquor overflow from this settling cone is further cooled in ammonia vacuum coolers to crystallize sal soda, which is filtered and converted to sodium carbonate monohydrate in double effect steam-heated vacuum evaporators. This product is again filtered and fed to oil-fired rotary driers. The calcined soda ash is stored in concrete silos at the rate of about 300 tons daily.

Lithium Production: One of the minor coproducts of the main evaporative cycle from a tonnage standpoint is lithium sodium phosphate. The product analyzes 21 pct Li_2O and constitutes the highest known grade of crude lithium material on the market. It is and will continue to be a major source of lithium for manufacture of lithium products.

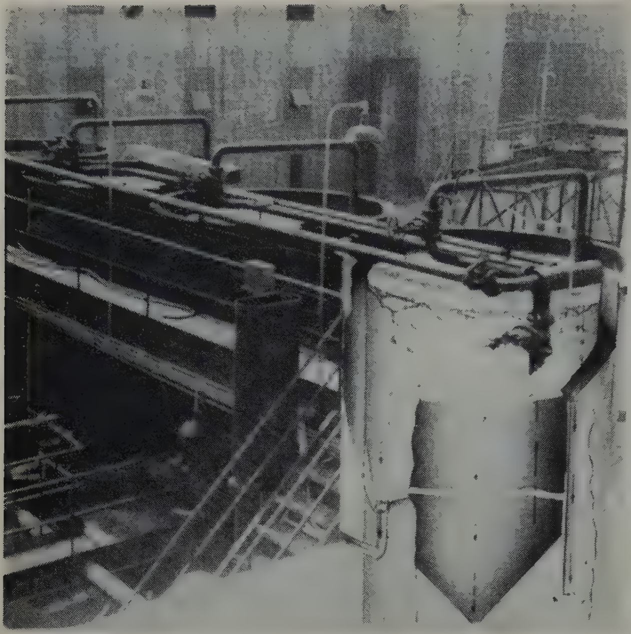

Fig. 5—Cross-section of one of four flotation tanks at lithium plant, showing inner tank construction, direction of liquor flows, and froth discharge.

The lithium content of the raw brine fed to the main plant evaporator cycle is 0.015 pct Li_2O. As a result of heating and concentration of the raw brine, the lithium is caused to precipitate mainly in the first effect evaporator as di-lithium sodium phosphate. The solid precipitate is extremely fine and averages 2.5 microns in particle size.

A fatty acid derivative is added in small amounts to the main evaporators to control foaming under vacuum concentration. This fatty acid saponifies in the hot alkaline medium, and the soap formed, together with the subsieve colloidal lithium material, becomes occluded by the inversely soluble burkeite precipitate. This material is freed from the burkeite at the burkeite dissolution step of the soda products process as previously outlined.

Thus, the lithium slime remains undissolved to a great extent in the burkeite liquor and is but slowly soluble, due, presumably, to its soap coating. In this condition the material is nonfilterable. It was once a troublesome and expensive plant problem, but an efficient method was devised for its recovery from burkeite liquor.

Up to 150 tons per month of this material are being recovered continuously by a unique flotation process. A light mineral oil is added to the burkeite dissolution tank. Aeration is introduced at the burkeite liquor cooling towers as the liquor is sprayed vertically downward through upward rising induced drafts of air. The air so introduced is emulsified by the soap and oil reagents present in the liquor as it is transferred via centrifugal pumps to the lithium flotation tanks.

Four 10,000-gal flotation tanks constitute the battery of flotation cells. A cross-section of one of these flotation tanks is shown in Fig. 5. These tanks are operated in parallel, stand 20 ft high by 9 ft diam, and are equipped with inner discharge cones and froth overflow launders. An automatic constant-level control device maintains constant head, which may be raised to yield froths of varying degrees of concentration. At other times, the level may be so adjusted as to rise at regular intervals and discharge the froth batchwise.

At the four flotation tanks, the burkeite liquor is allowed to become relatively quiescent. At the normal flows of 600 gal per min, it is relatively free of turbulence. Feed entry to the tanks is accomplished at the bases and discharge takes place by means of the inner cones. Each cone is tapered from 6 ft diam at the points of entry, 6 ft below the froth overflow lip, to 9-in. discharge at the base. As the liquor rises through the annular space surrounding the cone, the particles of soap-oil coated lithium compound contact extremely fine air bubbles and coagulate into floccules or clumps. These rise to the liquor surface in the form of a stable, dilute foam. The foam concentrates upon standing before it is discharged across the froth overflow lip. Clarified effluent from the flotation tanks is routed to the soda products sulphate refinery. This constitutes the lithium froth flotation process as developed at the Trona plant. The absence of an extraneous compressed air supply, the absence of moving parts, and the substitution for close attention through automatic control devices is extant. The proportioning of dilution and froth modifier reagent requirements in the burkeite liquor makeup step at the dissolver tank likewise is controlled by sensitive pneumatic density-controlled devices. These are made to operate under extreme variations of liquor flow rates.

The lithium foam is collected by gravity flow from the froth launders into receiver tanks. These tanks are equipped with circulating pumps and suitable steam heating equipment. By heating to a temperature of 150°F, the foam is de-aerated and the slurry is leached for sulphate impurities. It is then filtered batchwise on Sweetland filter presses. The filter cake is partially air-dried and then is put through steam-jacketed mixers for final drying to less than 2 pct moisture content. The dried concentrate is packed in 100-lb bags to await shipment.

This completes the main plant cycle of the American Potash and Chemical Corp. operations at Trona. The total production of high grade chemicals amounts to 650,000 tons annually. Trona continues to contribute high quality potassium chloride products for the farmer and the chemical manufacturer. Its output of soda ash as a basic heavy chemical and one of the most widely used industrial chemicals is being increased. Trona's borax and boric acid production, sold mainly to the glass and enamel industries, is steadily increasing. Salt cake is used principally in the kraft paper industry and in the manufacture of industrial and domestic detergents. Trona continues to be the major source of lithium for the manufacture of refined lithium products.

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Diesel Truck Haulage Through Inclined Adit

by V. C. Allen

THE Tri-State Zinc, Inc., Galena, Ill., was confronted with the problem of securing ore from a deposit because the hoisting shaft was several thousand feet from the mill. The orebody is several thousand feet long, averaging 200 ft in width and 60 ft in height and opened up by vertical shafts some 300 ft deep. Mining is by the room-and-pillar method. During the initial operation the ore was loaded by conventional electric ½-yd boom-and-dipper shovels and hauled to the shaft by 8-ton diesel trucks. This underground ore loading and hauling was well adapted to the conditions and productive of low costs per ton. However, with the mill situated as mentioned, a triple handling of all broken rock was necessary: 1—from the stope to the shaft by truck, 2—up the shaft by skip or can into the surface hopper, and 3—by truck from the surface hopper to the crushing plant at the mill. In addition to the repeated handling, serious troubles were encountered during the winter because of freezing in the shaft hopper.

Consideration was given to either moving the mill to the new orebody or to the construction of a second mill. The presence of other orebodies to be mined at a later date made the first alternative impractical while the capital outlay for a second mill, when the present plant of approximately 850 tons per day was deemed sufficiently large for the total reserves, made the second alternative also unwise. It was decided to retain the mill in the original location and continue to move the ore to it.

The idea of driving an inclined adit from the surface to the bottom of the orebody suitable for truck haulage and big enough to allow the passage of all mechanical equipment was conceived. Among the apparent advantages of such an incline were: 1—Direct haulage from the stope to the mill without rehandling. 2—Elimination of virtually all grizzlies. Trucking from underground to the mill would do away with all hoppers, chutes, gates, and skips and make the maximum rock size dependent solely on the size of the shovel dipper at the mine and the crusher opening at the mill. 3—Less secondary blasting would be needed. 4—Ease of transporting equipment and supplies underground. Shovels and trucks could be taken through the incline intact. 5—Equipment could be brought to the surface for repairs and servicing without loss of time. The same advantages of ease in moving would be present in the handling of men, steel, powder, and supplies. 6—There would be far less difficulty in increasing the amount of tonnage that could be moved by truck up an incline than would be found in attempting to increase the capacity of a shaft. 7—All the broken ore in the stopes would serve as bin capacity, as it would take the breakdown of all of the loading and hauling equipment to have the same effect as a delay in shaft hoisting. 8—All danger of men being trapped in the mine as a result of shaft fire or power stoppage would be eliminated. 9—Virtually all trouble from severe winter conditions would be eliminated by the direct haul underground to the mill.

The decision was made to proceed with the driving of an inclined adit. The topography of the surface between the orebody and the mill was such that it was possible to locate the portal at a point 170 ft above the mine floor and 1800 ft horizontally from the central point of the orebody to the south and 2500 ft from the mill to the north. A grade of 10 pct was found to be optimum for continuous truck haulage when the various factors of speed, safety, and truck maintenance were all considered. The incline as driven was consequently 1700 ft long on 10 pct grade and 12 ft high by 17 ft wide in cross section. The tunnel-driving equipment was chosen so that it could be used in mining after the completion of the tunnel. Drilling was done with a jumbo with two Joy jibs mounting 3-in. drills, loading with an Allis-Chalmers diesel-powered, front-end loader of approximately 1¼-yd capacity, and hauling by Koehring Dumptor trucks of 8-ton capacity, diesel-powered. The width of the tunnel allowed the end loader and Dumptor to be placed abreast. Since the Dumptors can be driven either forward or backward with equal facility, loading was accomplished without turning around either machine throughout the loading operation. The crew in addition to the tunnel foreman was comprised of three men per shift at the start and in the later work, four men. Each crew could perform any part of the working cycle. If the drilling was completed and the round blasted in the middle of a shift, the same men would proceed with the loading and hauling. Since the mine already had been drained to the bottom levels, no water was encountered. At the halfway point the tunnel was widened for approximately 100 ft to permit trucks to pass.

The total cost of the tunnel excluding the capital outlay for equipment, which was all continued in use in the subsequent mine operation, was $60,363.00 or $35.50 per ft.

The tunnel was completed at the end of June, 1949 and has been in continuous use since that time. In the five months from July to November inclusive, 106,049 tons have been transported to the mill or an average of 835 tons per day. No unforeseen disadvantages have been encountered and the advantages which had been predicated before the adit's construction have been more than realized.

As previously mentioned, the deposit is worked by the room-and-pillar system with occasional faces up to 125 ft high. Except in driving development drifts when diesel-powered, front-end loaders such as were used in the tunnel are employed, all shoveling is done by ½-yd boom-dipper type shovels electrically driven. These units need a width of 25 ft and a height of 14 ft in which to operate. All hauling is by diesel trucks, mainly Koehring Dumptors. Roads are maintained with caterpillar tractors and a road grader. The tonnage output from the

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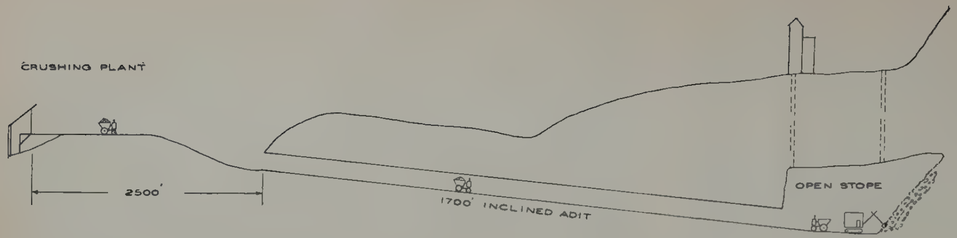

Fig. 1—Diesel truck haulage through inclined adit.

mine is limited by the capacity of the mill, which is approximately 850 tons per 24 hr. The mine is operated in two shifts; all drilling and blasting on the day shift, 7 a.m. to 3:30 p.m.; and loading and hauling on both the day and night shift, 7 p.m. to 3:30 a.m. The shifts are separated by a 3½-hr gap to allow powder smoke to clear. The loading and hauling crews on each shift are comprised of four men, one shovel operator and three truck drivers using three Dumptors similar to those used in the adit. In some instances when it is desired to get rock from several points simultaneously, two shovels are used. In such cases one shovel and operator will load two trucks, each with a driver. The other shovel operator will both load and drive the truck. The daily output of between 800 and 900 tons thus is loaded at the stopes and hauled to the mill crusher plant by a total of eight men or over 100 tons per man shift. The loading and hauling could be done on one shift with fewer men if there were sufficient hopper capacity at the mill.

The rock from the mine is dumped directly into a 40-ton hopper feeding into an apron feeder ahead of a 36x21-in. jaw crusher. The bin capacity between the coarse crushing plant and the mill proper is approximately 300 tons. There has never been any question of lack of mill feed as the haulage system is flexible to the demands of the mill. Broken rock stored in the stopes can be made immediately available to the mill. Conversely, any delay or shutdown in the mill has no effect on the underground operation except that the four men engaged in loading and hauling become idle. The occasional boulder too large to pass through the grizzly ahead of the apron feeder, bars set 20x30 in., are so few in number that they are handled by the truck drivers without additional help. All trucks are brought to the surface shop for daily servicing and repair work. Minor jobs on the loading equipment are made underground. The same applies to the jumbos, either tractor-mounted or truck type.

Since the mine is shallow and is opened by two shafts and the inclined tunnel, little mechanical ventilating aids are needed. One blower of 24,000-cfm capacity blows fresh air down a shaft, and one smaller blower forces fresh air down a drill hole. Since both shafts are covered, the incline serves as an upcast. Recent tests show 30,000 cfm discharging from the incline with the carbon monoxide content in the tunnel never exceeding 10 parts per million, far below the permissible amount. The aldehyde in the exhausts is barely noticeable in the adit itself despite the constant traffic of the Dumptors, a trip each way about every 5 min. Men are transported from the surface change house into the mine by a diesel truck with a removable housing and equipped with benches. This housing is necessary only during the winter months when the sudden change from the warm mine to occasional sub-zero surface temperature in wet mine clothes would be too severe.

Total mining costs including haulage are $0.9708 per ton. A partial breakdown of these costs follows:

Breaking	
Labor, including compensation and Social Security	$0.2564
Explosives	0.1451
Power	0.0317
Repairs and supplies	0.0652
Total	0.4984
Loading	
Labor, including compensation and Social Security	$0.0641
Power	0.0039
Repairs and supplies	0.0230
Total	0.0910
Hauling	
Labor, including compensation and Social Security	$0.0913
Diesel oil	0.0197
Repairs and supplies	0.0517
Road maintenance	0.0036
Total	0.1663
General	
Supervision, including compensation and Social Security	$0.0682
General labor	0.0584
Ventilation and lights	0.0074
Repairs and supplies	0.0811
Total	0.2151
Total overall costs	$0.9708

The results of the operation of the past five months have proved that mining by truck haulage through inclined or horizontal adits, especially in deposits of a type which permit loading into trucks at the ore faces, offers many advantages over vertical or inclined shafts.

In addition to the benefits derived in mining a single deposit, it also would appear that if a number of scattered orebodies are to be treated through a central mill much duplication of equipment and saving in manpower would result through the use of adits since it becomes easily possible to so arrange the various stages of the combined operation as to permit moving all the equipment from one mine to the other. In a similar operation under normal shaft hoisting, each mine or shaft would have to be separately equipped both with surface hoisting plant and underground units.

There is no apparent reason why the use of inclined adits would have to be confined to very shallow mines. When it is realized that driving a 10 pct incline with the type of equipment such as was used here is entirely similar to drifting, it becomes apparent that its total cost to reach a certain vertical depth is possibly no greater than would be entailed in sinking, to the same vertical depth, a three-compartment shaft equipped with underground skip pockets and with its necessary surface equipment including hoist and ore bins. There is undoubtedly a point at which the length of the haul up the adit would limit its practical use but no attempt has been made to determine this. Further experience and new types of equipment may permit considerably higher grades than 10 pct and thus lessen the length ratio between adit and vertical shaft.

Efficiency and Sharpness of Separation in Evaluating Coal-Washery Performance

by H. F. Yancey and M. R. Geer

Various criteria for evaluating the performance of coal washeries are discussed critically, with particular reference to ideas developed at the First International Conference on Coal Preparation held in Paris in June 1950. Criteria that relate only to the sharpness with which a cleaning unit separates coal from impurity are compared with those that reflect loss of coal or quality of products.

THE performance of coal-washing equipment, that is, the effectiveness with which coal and impurity are separated, is a subject of undisputed interest to all concerned with the preparation of coal. Through the years much has been written in both American and European literature on the performance of coal-washing equipment and its interpretation, but until recently no effort has been made to consolidate this wealth of information into a single, comprehensive treatment of the subject. A splendid effort in this direction was made at the First International Conference on Coal Preparation held in Paris in June 1950 in which representatives from virtually every coal-producing country participated. Although the scope of the conference covered the entire field of coal preparation, many of the sixty-odd papers and a great deal of the discussion dealt with the subject of evaluating washery performance. Prominence was given this subject by the Centre d'Etudes et Recherches, des Charbonnages de France, which sponsored the conference.

This research organization, generally referred to as Cerchar, was formed in 1948 following the nationalization of the French coal industry. Since that date it has conducted an intensive study of coal preparation. Perhaps because of a fresh viewpoint, Cerchar realized immediately that complete information on the performance of French washeries and a thorough understanding of the mechanism of gravity separation were the fundamental prerequisites in a program to improve preparation practice. For this reason Cerchar concentrated on obtaining performance data and on a comprehensive study of methods of evaluating such information. Some idea of the magnitude of this work will be appreciated from the fact that 175 performance tests were made in 44 plants.

The work performed by Cerchar emphasized a condition that is recognized by many of those concerned with coal preparation, namely that the lack of uniformity in presenting data on washery performance, the diversity of methods used in evaluating such information, and the frequent confusion of the terms related to performance have been prime factors militating against the progress from art to science. Thus, Cerchar feels that standardization of the methods employed for evaluating washery performance, not only in France but in the entire world, is essential to progress, and the prime objective of the Paris conference was to seek international standardization of the evaluation methods advocated by Cerchar. At the conclusion of the conference the question of standardization was referred to the International Standards Organization.

A careful study of the dozen papers on the subject of evaluating washery performance presented at the conference reveals that none of the methods is new; most of them date back over a decade. Nevertheless, the intensive scrutiny of these methods brought about by the conference unquestionably has resulted in a valuable exchange of ideas and a clarification of thought that could scarcely be achieved in any other way.

The object of the present report is to make available to the American coal industry the ideas developed at the conference which seem most important from an American viewpoint and to provide a critical analysis of some of the evaluation methods advocated by Cerchar. The various methods of evaluating washery performance are illustrated partly with data that have been published in American literature, and some new data have been obtained as the result of experimental work undertaken since the conference to provide information on certain controversial points. An English translation of the proceedings of the conference, including the discussion as well as the papers, will be available,[1] and previous to the conference Coal Age[2] published a very brief description of some of the principal evaluation methods used by Cerchar.

Americans will appreciate as well as do the French the desirability of standardizing both the termi-

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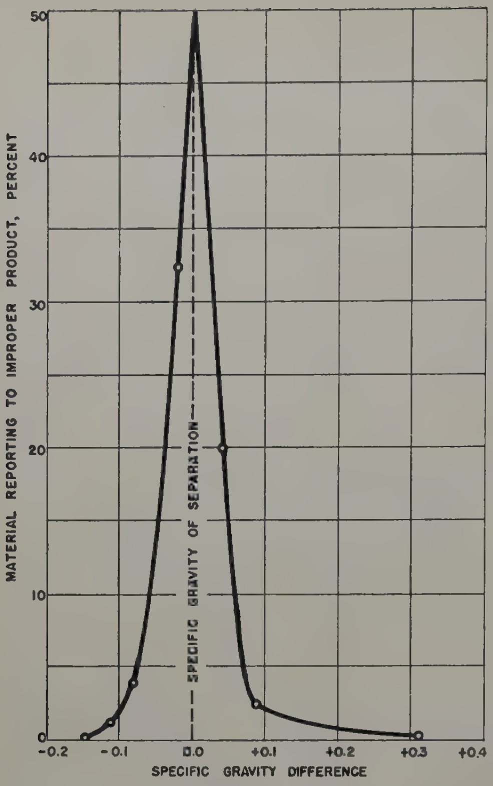

Fig. 1—Error curve for gravity separation.

nology and the methods of presenting washery performance data. However, the idea of rigid standardization will have less appeal here than in France where nationalization of the coal industry has brought centralized responsibility for all coal-preparation work.

Mechanism of Gravity Separation

The subject of evaluating washery performance can hardly be approached without the prelude of a brief statement concerning the mechanism of gravity separation. The ideal cleaning process, from the standpoint of performance, would be one in which all coal lower in density than a predetermined value would be recovered in the clean product, and all material of greater density would be rejected as refuse. Naturally, no cleaning process achieves this goal. The type of separation between coal and impurity which actually is accomplished may be illustrated best by Fig. 1.

Coal of low density and impurity of high density report largely to their proper products—washed coal and refuse, respectively. But as the density of separation is approached, the proportion of material reporting to an improper product increases rapidly. In fact, the density of separation can be defined best as the density of the material in the raw coal that is distributed equally between washed coal and refuse. The shape of this curve is like that of a Gaussian error distribution curve, and a mathematical analysis of gravity separation demonstrates that it is governed by the law of probability.

The imperfect separation of materials of intermediate density illustrated in Fig. 1 is characteristic to some degree of all coal-washing processes regardless of their mode of operation. The shape of the curve is determined largely by the inherent difficulty of stratifying materials differing only slightly in density. With processes that are capable of effecting a sharp separation between coal and impurity, such as heavy-medium units, the two limbs of the curve lie close together and approach or reach the 0 axis at densities not far removed from the density of separation. With processes that are unable to make a sharp separation between coal and impurity, the two limbs of the curve are more widely separated and approach 0 only at densities substantially removed from the density of separation. In some cases neither limb reaches the 0 axis.

It will be noted that the error curve shown in Fig. 1 is not symmetrical about the abscissa representing the density of separation but is skewed at higher densities. This skewness, which represents less perfect separation of the material of higher density, characterizes the error curves for most separations made at the usual washing densities.

The concept of representing a gravity separation as an error curve and defining the density of separation as the density of material that is divided equally between the two washery products dates back to a paper published by Tromp[3] in 1937. Terra[4] developed the probability concept of gravity separation along different lines in 1938.

Classification of Performance Criteria

The various criteria that may be used in evaluating washery performance may be classified according to the degree to which they are dependent upon the character of the raw coal being washed; some are directly dependent, others are much less so.

The performance of any washing unit depends upon three primary factors: the density composition of the feed, the density of separation, and the sharpness with which the unit separates coal from impurity. All three factors influence directly the quantity and quality of the washed coal and refuse, and hence any criterion of washery performance that is based on the quantity or quality of the products is influenced directly by all three factors. Thus, as is well known, the ash reduction effected by a washing unit, or the ash content of the refuse it produces, tell little about the inherent characteristics of the unit, for both values reflect the influence of the type of coal being washed and the density of the separation.

In the routine control of a washery, performance evaluation by means of the ash contents of the clean coal and reject or by testing these products on a control float-and-sink bath is entirely satisfactory. The composition of the raw coal does not vary greatly, and the density of separation is reasonably constant. Moreover, the quality of the washed coal and the yield are the factors of prime interest to the operator.

Often, however, the criterion of performance desired is one reflecting the inherent ability of the washing unit to make a sharp separation between coal and impurity, and this criterion must be as free as possible from the effect of either the character of the raw coal or the density of separation. Only in this way can the performance of washeries treating coals of dissimilar character and making separations at different densities be compared directly. Such criteria are useful in formulating performance guarantees for new plants and in calculating the yield and quality of products obtainable by treating a new coal in an existing plant of known performance characteristics. In other words, they assist in translating the washability data obtained from float-

and-sink tests into the yields and ash contents of a washing operation.

Dependent Criteria

The number of possible ways of utilizing some measure of the quality of the clean coal and refuse produced by a washer to characterize its performance are almost unlimited, and many such methods have been proposed through the years. Hence, the present discussion will be confined to only a few criteria that have found wide acceptance or for other reasons are of special current interest.

Efficiency: Efficiency formulas belong to the class of criteria that are influenced both by the character of the raw coal and the density of separation as well as by the sharpness with which a washing unit separates coal from impurity. A number of efficiency formulas have been proposed from time to time, but the one used most widely has been the Fraser and Yancey formula for recovery efficiency advanced by the Bureau of Mines in 1922.[5] This formula is the ratio, expressed in percentage, of the yield of washed coal to the yield of float coal of the same ash content shown to be present in the feed by specific-gravity analysis. It is written as follows:

$$\text{Efficiency} = \frac{\text{Yield of washed coal}}{\text{Yield of float coal of same ash content}} \times 100$$

Cerchar uses this formula, terming the value obtained "organic efficiency," and the discussion at the Paris conference disclosed the fact that it is the one criterion of washery performance that is universally recognized throughout the world. Doubtless the wide acceptance of the formula is attributable to its simplicity and to the fact that it conforms with the usual engineering concept of efficiency, that is, output divided by input.

Despite such wide use, some confusion regarding the Fraser and Yancey formula was evident at the Paris conference. Actually, two efficiency formulas, rather than one, were advanced in 1922. The second formula was written as follows:

$$\text{Efficiency} = \frac{\text{Yield of float coal}}{\text{Yield of washed coal}} \times \frac{\text{Raw-coal ash} - \text{washed-coal ash}}{\text{Raw-coal ash} - \text{float-coal ash}}$$

In this formula the yield of float coal and the float-coal ash refer to the product obtained by a float-and-sink separation at a specific gravity representing the most profitable separation of the coal. In practice, the most profitable separating density generally is difficult to determine, and hence this equation has not been used by the Bureau of Mines for many years. Moreover, depending upon the density selected to represent the most profitable separation, this formula may give values of over 100 pct.

The most serious shortcoming of the first or output-input efficiency formula, also called recovery efficiency, is that when no refuse is removed the efficiency is 100 pct. For example, pneumatic cleaning units treating unsized feed generally remove no refuse finer than about 28-mesh in size, and hence the efficiency for the treatment of the finest sizes in such units often is 100 pct, which is higher than the efficiencies obtained on the coarser sizes in which the ash reduction is much better. The significance of such an efficiency value is that it represents no loss of clean coal in these sizes rather than indicating perfect separation.

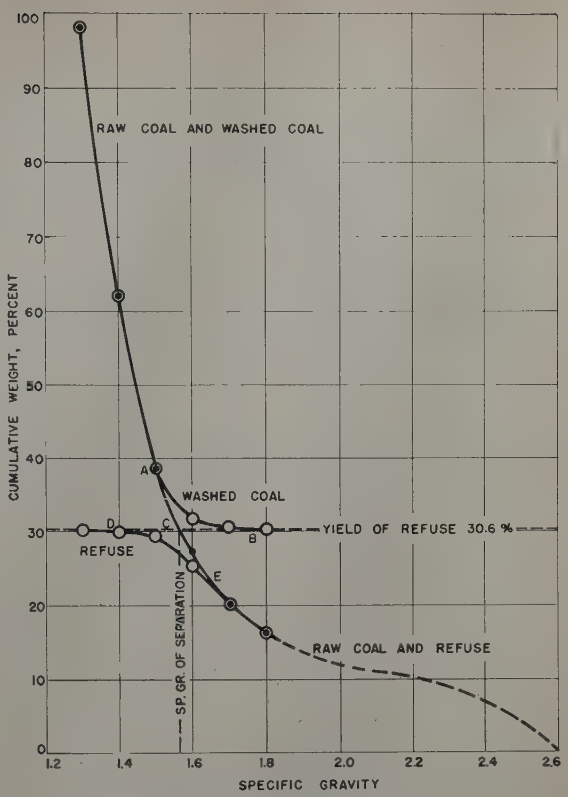

Fig. 2—Yield-specific-gravity curves for raw coal, washed coal, and refuse plotted to show "triangles of error."

Use of the efficiency formula requires only a specific-gravity analysis of the raw coal and the ash content of the washed product, and in many cases the calculation of efficiency from these data is entirely satisfactory. However, with a friable coal, and more particularly with one containing intergrown particles from which coal is freed readily by degradation, it is preferable to calculate efficiency from the composite specific-gravity analysis of the washery products. In this way the effect of degradation in the washing process is eliminated, and the washing unit is not given credit for the recovery of particles of coal that were not present as such in the feed. In rare instances efficiencies of over 100 pct have been reported as the result of degradation when the specific-gravity analysis of the raw coal rather than a composite specific-gravity analysis was used.

The field of application of efficiency values is clearly defined. Since efficiency is influenced by the character of the raw coal being treated and the density of separation as well as by the inherent characteristics of the washing unit, washers treating coals of different density composition or operating at different densities of separation cannot be compared directly on the basis of this criterion. In a given cleaning unit, a coal containing a high proportion of intermediate-density material cannot be cleaned with as high efficiency as a coal containing little material of intermediate density. Similarly, a separation at a lower density cannot be made with as high efficiency as a separation at higher density.

Misplaced Material: One of the simplest, and certainly one of the most obvious, evaluations of washery performance is the quantity of sink material in the washed coal and float material in the refuse. If the washery products are tested at the density at

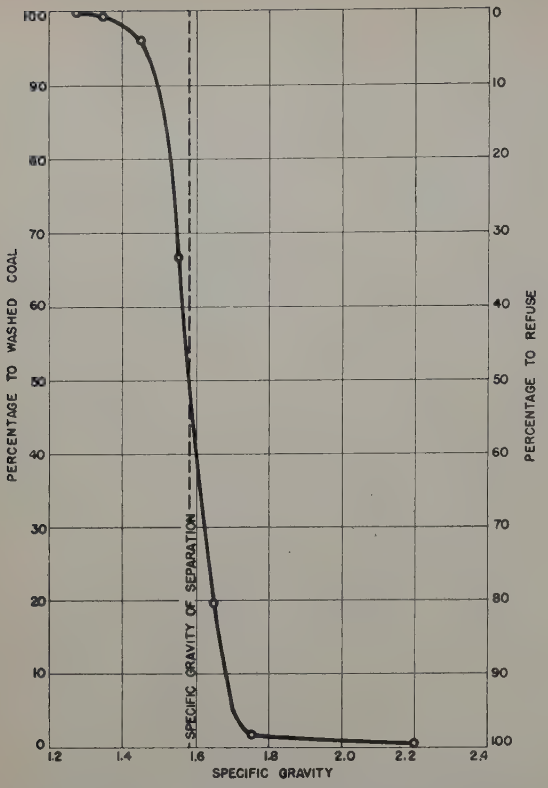

Fig. 3—Distribution curve for heavy-medium separation in 2-ton per hr pilot plant.

which the washing unit is operated, the sink in the washed coal and float in the refuse represent material that has been misplaced. An early example in the American literature of the concept of misplaced material as an evaluation of washery performance is found in the work of Lincoln.[6] The use of the quantity of misplaced material as a criterion of washery performance was employed by the Bureau of Mines in 1938,[7] and in 1950 Anderson[8] proposed the use of this criterion as an efficiency value to replace the Fraser and Yancey formula.

British literature also contains examples of this criterion, and in 1947 R. T. Hancock[9] defined the density of separation as being the density at which the total quantity of misplaced material is a minimum. Cerchar likewise has employed the quantity of misplaced material as a criterion of washery performance, using the term "error by strays" for this quantity. Cerchar has concluded, somewhat at variance with Hancock, that at the density of separation the amounts of sink in the washed coal and float in the refuse, when expressed as percentages of the feed, are equal, and the error by strays is defined as either the amount of sink material in the washed coal or float in the refuse at the density of separation.

Fig. 2 illustrates a type of plot originally developed by Heidenrich[10] and now used widely by Cerchar. The curve for raw coal represents the cumulative percentages of sink material. The refuse curve also is plotted as cumulative sink, the percentages being expressed in terms of raw coal. The curve for washed coal represents the specific-gravity composition of the washed coal expressed as a percentage of the feed and is plotted as cumulative percentage float. To place the washed-coal curve in proper relation to the curves for raw coal and refuse, the cumulative float values are plotted as additions to the percentage yield of refuse. In other words, the cumulative percentage of float in the washed coal is plotted from an axis representing the yield of refuse. The curves shown in Fig. 2 are plotted against specific gravity, but similar curves are also obtained by plotting against ash content.

This diagrammatic representation of the results of a washing operation, while perhaps not easy to grasp because of the method of plotting, has the merit of showing in a pictorial way the type of separation obtained in a washing unit. The washed-coal and refuse curves coincide with the raw-coal curve in the ranges of low and high density, but diverge from it as they approach the density of separation. The density of separation obtained from this type of plot, incidentally, is not the same as that previously defined as the density of material that is divided equally between clean coal and refuse; it is simply the density of the float-and-sink bath that would be required to give a yield of float coal equalling the yield of clean coal obtained in the washing operation. Moreover, this value for the density of separation differs from the value frequently defined in American practice as being the density of the float-and-sink bath that would yield a float product having the same ash content as the washed coal.

The divergence of the washed-coal and refuse curves from the raw-coal curve forms "triangles" ABC and DCE. These triangles are termed "triangles of error" by Cerchar, and it is obvious that their area is related to the quantity of misplaced material and therefore can be used to characterize the quality of the gravity separation.

Both the area of the triangles of error and the quantity of misplaced material are, like efficiency, influenced by the character of the raw coal, the density of separation, and the sharpness with which the washing unit separates coal from impurity. In fact, Cerchar employs a method of calculating efficiency from the area of the triangles of error, and the efficiency value obtained in this way is in many cases the same as that obtained by using the ratio of washed coal to float coal employed in the Fraser and Yancey formula.

Independent Criteria

As early as 1912 David Hancock[11] evaluated washery performance by showing graphically the proportions in which each density fraction of raw coal was distributed between clean coal and refuse, but 25 years elapsed before Tromp[3] plotted these proportions against density to obtain a distribution curve. This type of curve, which was also developed

Table I. Specific-Gravity Analyses and Distribution Data

	Specific-Gravity Analyses, Pct of Product			Specific-Gravity Analyses,[a] Pct of Feed		Distribution,[b] Pct		
Specific Gravity	A Feed	B Washed Coal	C Refuse	D Washed Coal	E Refuse	F Feed	G Washed Coal	H Refuse
Under 130	1.7	2.5	0.0	1.7	0.0	100.0	100.0	0.0
1.30 to 1.40	36.3	52.2	0.3	36.2	0.1	100.0	99.7	0.3
1.40 to 1.50	22.9	31.8	2.6	22.1	0.8	100.0	96.5	3.5
1.50 to 1.60	11.8	11.3	12.8	7.9	3.9	100.0	66.9	33.1
1.60 to 1.70	6.6	1.9	17.2	1.3	5.3	100.0	19.7	80.3
1.70 to 1.80	4.5	0.2	14.2	0.1	4.3	100.0	2.2	97.8
Over 1.80	16.2	0.1	52.9	0.1	16.2	100.0	0.6	99.4

[a] Column D obtained by multiplying column B by 69.4 pct, the yield of washed coal; column E obtained in corresponding manner.
[b] Column G obtained by dividing column D by column A.

independently by the Bureau of Mines in 1938,[12] provides means for evaluating the sharpness with which a washing unit separates coal from impurity. The evaluation is substantially independent of either the character of the raw coal or the density of separation and thus provides information that is related to the inherent characteristics of the washing unit. Thus, the distribution curve and various criteria that can be derived from it differ fundamentally from the criteria described previously.

As an illustration of the distribution-curve method of expressing washing results, Table I gives specific-gravity analyses of the feed, washed coal, and refuse from a test made in a laboratory, heavy-medium, pilot plant of 2 tons-per-hr capacity. The distribution data are shown in the table, and the curve in Fig. 3. In constructing the curve, the distribution data for each density fraction are plotted against the mean density of the fraction, the mean density being the density at which half of the material would float and half would sink. Assumptions are required in plotting the lightest and heaviest fractions. Material lighter than 1.30 sp gr is plotted at 1.28, but any errors in assuming the mean density of this fraction generally have little influence on the shape or the position of the curve. The fraction heavier than 1.80 sp gr is plotted at 2.20 or 2.30, depending upon judgment concerning the character of this material; the validity of this assumption often has considerable influence on the shape of the curve. Generally it is sufficiently accurate to plot the data for fractions of intermediate density at the arithmetic average of the limiting densities.

Fig. 3 indicates that none of the lowest gravity coal was lost in the refuse product, and almost no impurity heavier than 1.80 sp gr entered the washed coal. The specific gravity at which the curve crosses the 50 pct weight ordinate, 1.59, is the specific gravity of separation; material of this density was divided equally between washed coal and refuse. A distribution curve representing a perfect separation would show all coal lighter than the density of separation going to the washed product and all heavier material going to refuse, with the curve passing from 100 pct to 0 on the abscissa representing the density of separation. The degree to which the distribution curve for a washing operation approximates this theoretical or perfect shape is a measure of the sharpness of the separation made by the unit.

The distribution curve generally is plotted directly against density as shown in Fig. 3, but for the purpose of comparing distribution curves representing separations at different densities, the data can be plotted against the difference between the mean density of the fraction and the density of separation. The relationship between a curve plotted in this way and the plot shown in Fig. 1 becomes evident if the portion of the curve in Fig. 3 for densities less than the density of separation is revolved 180° about the 50 pct ordinate.

Error Area and Probable Error: The distribution curve is a convenient means of showing the sharpness of the separation effected by a washing unit and has found increasing use both in America and Europe. However, curves are a less convenient means of comparison than figures, and therefore several numerical criteria have been derived to characterize with a single figure the magnitude of the deviation of the distribution curve from the curve that would represent a perfect gravity separation. One such criterion is the area lying between the distribution curve and the lines representing perfect separation; this area has been termed "error area." If distribution curves are plotted to a fixed scale, the error areas provide a convenient means of comparing sharpness of separation. Driessen[13] advocated the use of 1 cm for each 2 pct of weight on the ordinate scale and for each 0.1 in specific gravity on the abscissa scale, with the area expressed in square centimeters. Cerchar also has adopted this scale. The same numerical result is obtained if inches are substituted for centimeters in plotting and the area is determined in square inches. Thus the error areas given in this report are directly comparable with those reported in European literature.

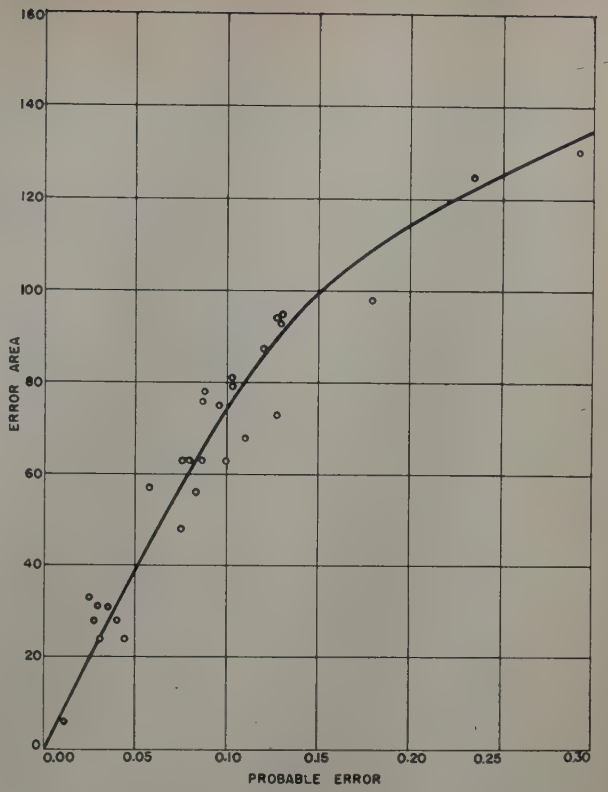

Fig. 4—Relation between error area and probable error.

As another method of characterizing the distribution curve with a single figure Cerchar has adopted a criterion originally developed by Terra[4] called "écart probable," which is variously translated as probable error or probable deviation. Numerically, the probable error is 50 pct of the difference between the specific gravities at which the distribution curve crosses the ordinates representing 25 and 75 pct recovery in the washed product. Thus, if the distribution curve is a straight line in the range between these ordinates, and many curves do approximate a straight line in this range of density, the probable error is simply a measure of the slope of the curve.

Both error area and probable error are used to indicate sharpness of separation and Cerchar expresses the relation between them mathematically. However, as shown by the plot of these two criteria in Fig. 4 for about 30 washing tests, the relation is only a general one. The choice of which criterion to use probably is largely a matter of individual preference. Probable error has the advantage of being more easily determined than error area, for the latter requires the use of a planimeter. On the other

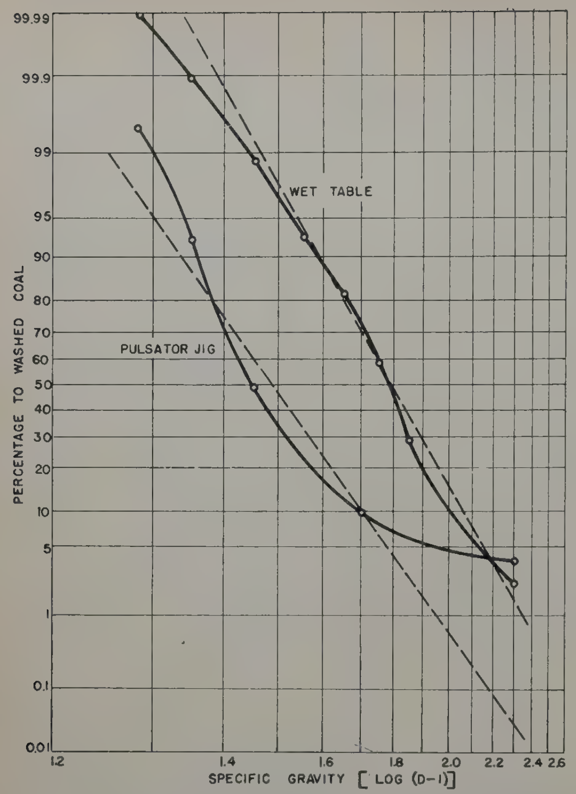

Fig. 5—Distribution curves plotted on logarithmic-probability scales.

hand, error area takes into account the entire distribution curve, while probable error evaluates only the portion of the curve in the range of intermediate density. Probable error thus does not reflect directly the disposition made by the washer of the lightest and heaviest fractions of the raw coal, and in most cases the disposition made of these fractions is of far more practical importance than the treatment of the intermediate-density material. Despite this criticism, however, Fig. 4 demonstrates that a general relationship between the two criteria does exist.

Distribution Curve Transformation

The advantages which accrue when a relationship can be plotted as a straight line are well appreciated: two parameters serve to describe the relationship completely, fewer data are required to obtain the same degree of accuracy, errors are more conspicuous, and interpolation is facilitated. With these objectives in mind, Cerchar advocates plotting distribution curves in straight-line form. Since as has been pointed out, a gravity separation can be regarded as a probability function, the ordinates of the distribution curve can be plotted on a probability scale to convert the curve to an approximation of a straight line. The scale of abscissa required to give the closest approach to a straight line is less easily defined, however.

Over eight years ago Vissac[14] submitted a paper to the AIME in which he advocated plotting distribution curves as straight lines, but this portion of his paper was not published; he employed an abscissa scale of log (d—d$_o$), d being density and d$_o$ being the density of the pure coal. Tromp,[15] on the other hand, employs a logarithmic scale of density raised to the 0.6 power. Cerchar uses three separate abscissa scales, depending on the type of washing unit. An arithmetic scale is employed for plotting the results of heavy-medium separations, a scale of log (d—1) is used for processes employing water as the separating medium, and a scale of log d is employed for pneumatic processes.

Fig. 5 shows two distribution curves plotted on a chart having a probability ordinate scale and an abscissa scale of log (d—1). One curve represents the operation of a pulsator-type jig treating a feed of 3 to 1¼-in., and the other is for the operation of a wet table on 3/32-in. to 0 coal. Neither curve is a straight line, but in the case of the table operation the deviation from straight-line form is not great; a maximum error of about 5 pct in weight would be incurred in plotting this curve as a straight line. The data for the pulsator jig, however, show greater curvature, and if plotted as a straight line would involve errors in weight of about 10 pct.

The authors have plotted a large number of distribution curves on the scales advocated by Cerchar, with varying degrees of success. The data always show some curvature, and in many cases plotting as a straight line involves a significant sacrifice in accuracy. The reports of Cerchar contain a great number of distribution curves plotted on a probability scale, but in most cases the accuracy of the straight-line representation cannot be judged because the plotted points are not shown and no numerical data are given. Even though Cerchar advocates the use of the straight-line type of plot, it admits that no distribution data form a perfect straight line and that in many cases the deviation is serious.

If a relationship has been definitely established as being a straight line, the advantages of a straight-line plot can be realized with confidence. In the case of the distribution curve, however, so many deviations from straight-line form have been observed that it does not seem feasible to use the straight-line relationship as a basis for reducing the number

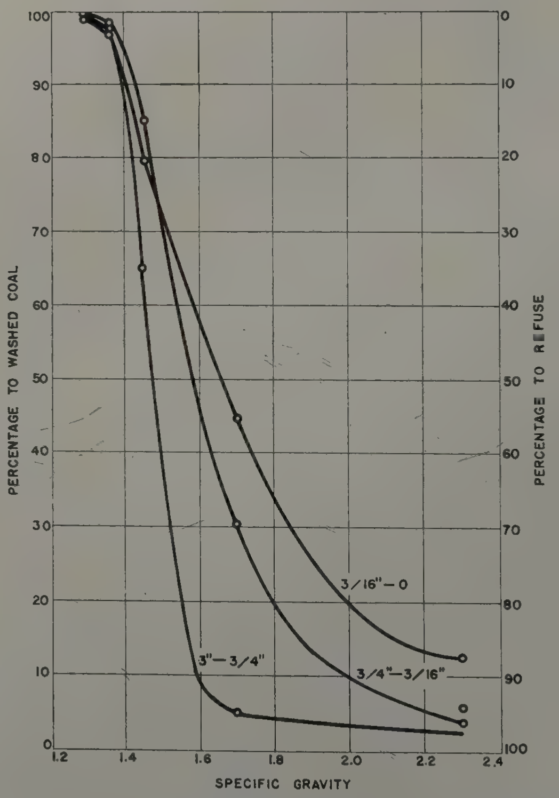

Fig. 6—Distribution curves for various sizes of coal cleaned in Baum-type jig.

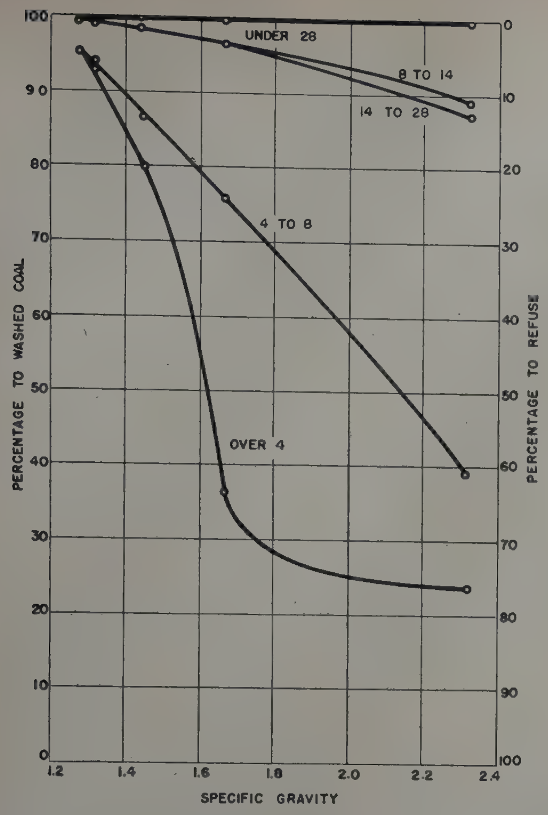

Fig. 7—Distribution curves for various sizes of coal cleaned on pneumatic table, sizes in mesh.

of densities used in performance tests. Likewise, since reliable data frequently deviate from a straight line, the relationship cannot be used to detect errors in experimental work. For these reasons, coupled with the fact that the use of a probability plot is a complication, the authors continue to plot distribution curves in arithmetic form. The arithmetic plot appears to be as useful as the straight-line form, and its greater simplicity renders it more suitable for general use in the United States.

Examination of Distribution Curve

The distribution curve and its related criteria, error area and probable error, are valuable tools, but they certainly do not represent a panacea for all the difficulties encountered in evaluating washery performance. They are subject to certain limitations which must be understood thoroughly to prevent their misuse.

One difficulty that is sometimes encountered in the use of the distribution curve is the amount of data required to fix accurately the shape and position of the curve. In American practice generally only four or five float-and-sink baths are employed in testing, and sometimes the data available for evaluating performance represent tests on only two or three densities. Occasionally the shape of the distribution curve is fixed with acceptable accuracy by only four points, but more frequently some latitude exists in drawing the curve even when five or six densities have been used in testing. The greatest difficulty in plotting the curve correctly occurs when the samples have not been tested at specific gravities higher than 1.60 and an appreciable proportion of the heaviest fraction has entered the washed product. Under these conditions it is virtually impossible to locate the curve exactly in the range of higher densities, with the result that the error area cannot be determined with complete accuracy.

European practice apparently involves the use of more and higher densities in examining samples for performance tests. Thus, American practice may have to follow the European system if distribution curves are to be more generally used. The amount of data required to give an accurate distribution curve constitutes one of the most serious limitations in the use of the curve. The same limitation applies to the efficiency formula advanced by Anderson[8] in 1950. His formula, which is the complement of the percentage of misplaced material, involves the use of a distribution curve for determining the density of separation—this being the density at which the percentage of misplaced material is calculated. To obtain comparable accuracy, use of this formula requires more data than that needed for the Fraser and Yancey formula.

Influence of Density Composition of Raw Coal: All those who have used the distribution curve have made the basic assumption that the curve was independent of the density composition of the raw coal. This assumption appears to be warranted because the method of calculating the distribution is not dependent upon the quality of the washed products. However, no data have ever been presented to show that a washing unit will yield the same distribution curve when treating coals of dissimilar density composition. Data of this type generally are not available.

The Bureau of Mines recently has undertaken an investigation in which coals of entirely different density composition will be washed in several different types of cleaning units to determine how much influence this factor has on the shape of the distribution curve. The problem is complicated by the fact that related factors such as particle shape and size composition must be comparable.

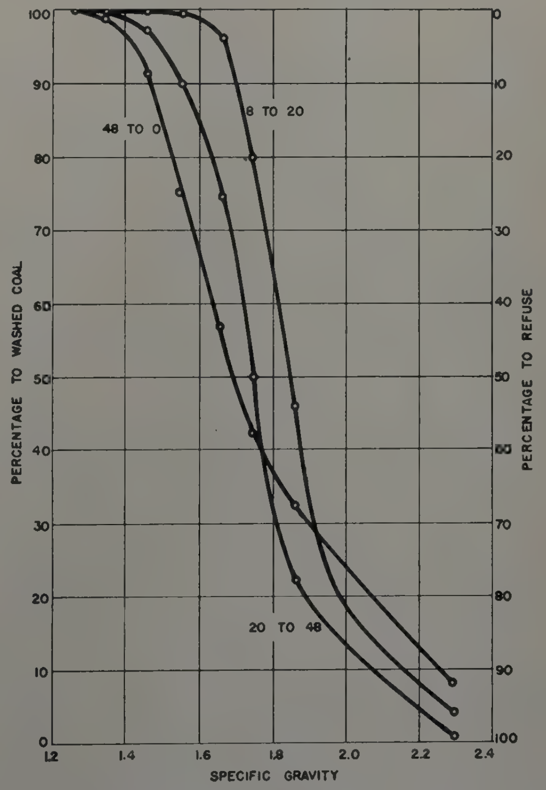

Fig. 8—Distribution curves for various sizes of coal cleaned on wet table, sizes in mesh.

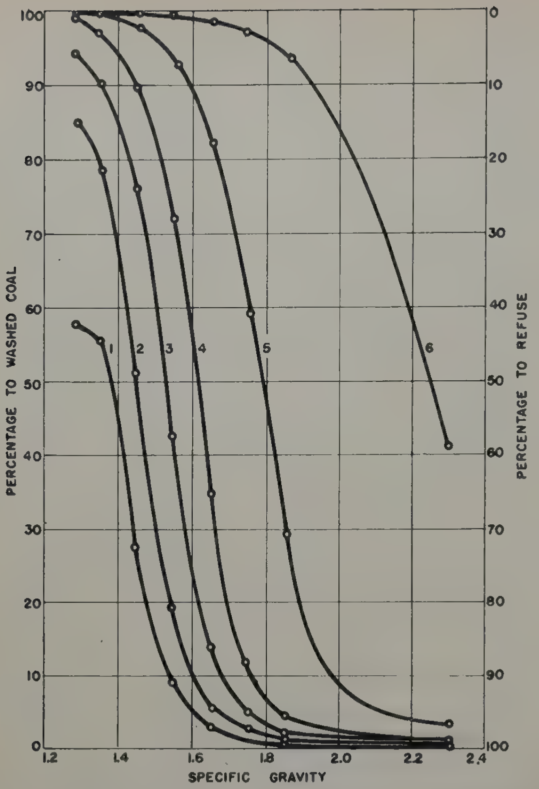

Fig. 9—Distribution curves for separations at various densities on wet table.

In this connection, it seems scarcely necessary to point out that the distribution curve no less than efficiency is influenced by such obvious factors as throughput, the mechanical condition of the washing unit, and the skill with which it is operated. Sharpness of separation is necessarily impaired by either excess feed rate or faulty operation.

Influence of Size Composition of Raw Coal: It is generally recognized that particle size modifies the influence of density in any type of cleaning unit, regardless of its mode of operation. All gravity concentrators employ currents of the separating fluid, and the size of a particle as well as its density determines its motion in the fluid. Thus, although the distribution curve for a gravity separation reflects primarily the influence of density, it cannot be entirely free from the influence of particle size as well. Even in continental European practice, where close presizing ahead of individual washing units is the rule, the influence of particle size on the shape of the distribution curve is well recognized.

The separation effected by a cleaning unit operating on a wide range of sizes is portrayed by a single distribution curve, but the performance of such a unit is shown much more completely and accurately by the family of curves that is obtained by plotting separately the distribution data for each individual size in the raw coal. Families of curves plotted in this way are shown in Figs. 6, 7, and 8 for the operation of a Baum-type jig, a pneumatic table, and a wet table. It will be observed that with each of these units the various sizes of coal were treated quite differently. Each size was separated at a different density, and each was characterized by different recoveries of coal and rejections of refuse.

Since particle size has a relatively greater influence in pneumatic units than in wet processes, the curves in Fig. 7 for the operation of the pneumatic table offer a good illustration of the importance of particle size in determining the shape of the distribution curve. The curves for the individual sizes treated in this unit indicate a marked decrease in sharpness of separation with decrease in particle size. In fact, no cleaning was accomplished in the sizes finer than 28-mesh. With the individual sizes being treated so differently, it is evident that the composite curve representing the operation as a whole is influenced directly by the proportions in which the sizes occur in the raw coal. If another coal of substantially different size composition were treated on the same table, the composite curve which would result would necessarily be substantially different from the one representing the test illustrated in Fig. 7. For example, if another coal containing twice as much material finer than 28-mesh were treated on the same table, the resulting composite curve for all sizes combined would show a separation that was distinctly less sharp.

All of the performance data studied by the authors, whether for wet, pneumatic, or heavy-medium processes, have shown the same trend in the relationship between particle size and sharpness of separation—the coarser sizes of the feed are always separated more sharply than the finer material. For wet processes the impairment in sharpness of separation with decreasing particle size is less pronounced than that observed for pneumatic units, and heavy-medium processes would be expected to show the least influence of particle size because they treat a limited range of sizes and do not depend upon currents to effect the separation between coal and impurity. However, so little detailed performance data for heavy-medium plants have been published that their performance characteristics are not well established.

A comparison of the influence of particle size in the operation of different types of washing equipment is evident in the curves of Figs. 6 and 8. In Fig. 6, representing the operation of a Baum-type jig, the distribution curves show an increase in the density of separation with decrease in particle size. This relationship between density of separation and particle size has been evident in nearly all of the performance data for jigs examined; in fact, Vissac[14] has derived an equation relating these two factors for jig operation. The distribution curves in Fig. 8 for the operation of a wet table, however, show the opposite trend. With the wet table, decreasing particle size is accompanied by decreasing density of separation. Some other data for table operation show a similar trend, and thus it may be concluded that with these

Table II. Specific-Gravity Analyses for Various Zonal Products from a Table Test, Pct

	Zone or Separation Number						
	1	2	3	4	5	6	7
Specific gravity							
Under 1.30	24.0	22.8	11.7	7.4	1.0	0.0	0.0
1.30 to 1.40	61.1	51.3	37.3	29.2	8.6	0.2	0.0
1.40 to 1.50	11.5	19.9	29.6	21.5	13.0	1.6	0.0
1.50 to 1.60	2.2	4.7	16.4	25.9	17.9	4.0	0.1
1.60 to 1.70	0.5	0.7	4.0	12.0	26.8	6.4	0.2
1.70 to 1.80	0.3	0.2	0.7	2.8	18.9	9.7	0.7
1.80 to 1.90	0.1	0.1	0.2	0.7	8.5	14.3	1.3
Over 1.90	0.3	0.3	0.1	0.5	5.3	63.8	97.7
Weight in zone, pct	31.0	15.0	10.6	8.4	8.5	13.1	13.4
Yield of washed coal, pct	31.0	46.0	56.6	65.0	73.5	86.6	

Table III. Relation of Performance Criteria for Various Separations on a Wet Table

Criterion	Separation No. 1	2	3	4	5	6
Yield of washed coal, pct	31.0	46.0	56.6	65.0	73.5	86.6
Ash in washed coal, pct[a], [b]	6.4	7.3	8.6	9.8	12.4	19.6
Ash in refuse, pct[a]	36.4	44.0	51.8	59.8	68.6	76.7
Efficiency, pct	57.0	78.4	88.3	95.6	98.0	99.0
Misplaced material, pct	24.3	16.7	11.2	6.5	4.4	
Specific gravity of separation	1.38	1.45	1.53	1.62	1.78	2.24
±0.10 sp gr, pct	57.5	27.6	16.9	11.4	6.7	
Error area	55	52	52	51	63	62
Probable error		0.080	0.075	0.070	0.075	

[a] Moisture-free basis.
[b] Ash content of raw coal, 27.3 pct.

two types of washing equipment the relationship between density of separation and particle size often is reversed. Distribution curves provide interesting comparisons such as this, which frequently are obscured when performance is evaluated in terms of ash content.

Influence of Density of Separation: Much of the usefulness of the distribution curve, and its related criteria error area and probable error, naturally depends upon the extent to which it is independent of the density of separation. In many cases it is desirable to compare the performance of washing units operating at different densities, or to project the known performance of a unit working at one density to the performance that may be expected when it is operated at a different density.

Most of those who have used distribution curves in evaluating washery performance have contended that the same washing unit will have different distribution curves when operated at different densities. Vissac,[14] Tromp,[15] and Cerchar appear to be agreed on this point; all of them have stated that probable error increases with increase in the density of separation. The authors, however, have felt that the distribution curve should be substantially independent of the density of separation, at least for densities within the range in which washing equipment usually is operated. None of the investigators mentioned above who contend that probable error increases with increase in density of separation have presented any data to demonstrate the relationship, except Vissac, whose data were not free from the influence of particle size. Vissac cited data showing that with the various sizes of coal treated in a jig handling an unsized feed the separations made at higher densities were characterized by greater probable errors. However, the separations made at the higher densities were those made on the finer sizes of the feed, and thus, particle size probably had more influence on the probable error than did density of separation.

To obtain data that would serve to clarify the question of whether probable error increases with increase in density of separation, a test was made on 3/32-in. to 0 coal from the Great Falls field, Montana, using a 4-ft laboratory wet table. The discharge of the table was sampled as seven zones, numbered from one to seven with successive increase in distance from the head-motion end of the table. Specific-gravity analyses were made on the individual zonal samples, using seven baths in the range from 1.30 to 1.90 sp gr. A table lends itself particularly well to an investigation of this type, because with a single test the data for the individual zones can be added together in a number of combinations to provide separations at various densities. That is, Zone 1 can be considered the washed coal, with Zones 2 through 7 constituting the refuse, or Zones 1 and 2 can be considered the washed coal and the remaining zones the refuse; with seven zones, six different separations are possible.

Table II gives specific-gravity analyses of the individual zones, and the weight recovered in each zone. The distribution curves plotted from these data by assuming the six different successive separations are shown in Fig. 9.

Table III provides a comparison between the various criteria that may be employed to characterize the performance of the table at the six different densities of separation. The inclusion of successive zones as washed coal increased the yield of this product from 31 to 86.6 pct, with a corresponding increase in the ash content from 6.4 to 19.6 pct. The separations obtained ranged from 1.38 to 2.24 sp gr, but both probable error and error area remained practically constant.

Separations one, two, and six do not represent normal washing practice; the yield of washed coal was abnormally low for separations one and two, and in separation six not enough of the heavy impurity was removed. Hence, the distribution curves for these three separations are not the type normally encountered, and the corresponding error areas are of questionable significance. Tromp[15] contends that error area is not a valid criterion to employ unless both ends of the distribution curve approach the 0 axis closely, and this contention appears justified. Thus, the relationship between probable error, error

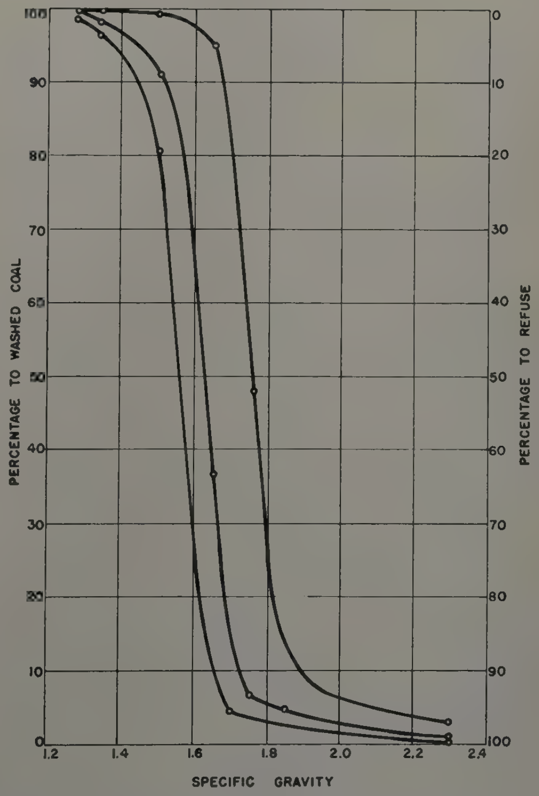

Fig. 10—Distribution curves for separations at different densities in Driessen cyclone.

area, and density of separation shown in Table III for separations one, two, and six probably should be regarded as of uncertain meaning. However, the distribution curves for separations three, four, and five represent types normally encountered, and the relationship between the various criteria for these separations is valid. Thus, on the basis of this table test, both error area and probable error can be regarded as being substantially independent of the density of separation in the range of separating densities at which coal-washing equipment generally is operated.

A similar conclusion can be drawn from the distribution curves shown in Fig. 10 for the operation of a Driessen cyclone as a heavy-medium cleaning unit. In the three tests represented by these curves, successively higher densities of separations were obtained by removing smaller proportions of refuse. The three curves are parallel, straight lines through the range of 75 to 25 pct recovery in the clean coal, and hence the probable error for all three separations is the same. Similarly, the error area for the separations, which were made at specific gravities of 1.56, 1.63, and 1.75, were 37, 41, and 43, respectively. Variations in error area of this magnitude are scarcely significant, and thus these tests, like the wet-table test previously described, indicate that the criteria probable error and error area are not influenced by the density of separation, at least in the usual range of separating densities.

Comparison of Criteria

Throughout the present report a distinction has been maintained between the sharpness of the separation effected by a washing unit and the efficiency of the washing operation. The two criteria are absolutely different, though sometimes confused because their relationship is not fully appreciated. For example, in "Coal Preparation,"[16] the term efficiency is used for the criteria which evaluate only sharpness of separation. This practice scarcely can be called improper in the absence of any rigid standardization of terminology. Nevertheless, current usage by Cerchar, Tromp, Driessen, Vissac, and others who are employing the concept of sharpness of separation preserves a distinction between it and efficiency. American practice should be no less discriminating.

Examination of the data in Table III, which summarizes the performance of the wet table, will clarify the influence of sharpness of separation on the more familiar criterion, efficiency. The efficiencies of the various separations made on the table ranged from 57.0 to 99.0 pct, even though the sharpness of separation—as measured by either error area or probable error—remained practically constant. Progressive increase in the density of separation was accompanied by a progressive increase in efficiency. Thus, equally sharp separations may provide entirely different efficiencies, depending upon the densities at which they are made.

The data in Table III show that as the specific gravity of separation increased there was less material within the limits of ±0.10 of this specific gravity. This decrease in the amount of material close to the density of separation generally has been regarded as being responsible for the improved efficiencies obtainable with separations at higher densities. Such a contention seems reasonable, since it is the material within ±0.10 of the specific gravity of separation that is treated with the least degree of sharpness. However, as stressed by Cerchar, the mix-up of material that occurs near the density of separation results in trading a small amount of material that should have been recovered in the clean coal for an equal amount of material that should have been rejected as refuse. These two groups of material differ little in ash content, and hence their improper distribution does not change the yield and usually has only a minor effect on the ash content of the clean coal; similarly it does not result in a great loss in efficiency.

Efficiency is influenced to a much greater extent by the recovery effected in the lower density range, for the loss of a few percent of this material gen-

Table IV. Comparison of Performance Criteria for a Number of Washing Operations

Cleaning Unit	Size of Coal, Inches or Mesh	Specific Gravity of Separation[a]	±0.10[b]	Error Area	Probable Error	Misplaced Material, Pct	Efficiency,[c] Pct
Heavy medium	Over ¼	1.56	7.0	6	0.011	1.4	
Heavy medium[d]	1¼ to ⅜	1.50	4.8	24	0.031	0.7	99.9
Heavy medium[d]	2 to ¼	1.57	22.5	28	0.040	5.8	97.1
Heavy medium	1⅝ to 15/16	1.79	20.3	48	0.074	3.5	
Heavy medium[d,e]	14 to 35	1.63	1.7	24	0.044	0.4	99.9
Heavy medium[d,e]	8 to 200	1.65	10.4	28	0.027	3.0	98.5
Jig, Baum	5 to 0	1.57	7.0	63	0.087	4.8	97.5
Jig, Baum	5 to 0	1.46	14.4	68	0.111	21.0	72.2
Jig, Baum	3 to 0	1.67	10.0	76	0.086	8.7	96.1
Jig, Baum	3 to 0	1.53	5.8	81	0.103	3.4	98.0
Jig, pulsator	3 to 1¼	1.45	19.8	57	0.057	7.9	94.0
Jig, pulsator	1¼ to ¼	1.70	2.3	95	0.130	2.2	99.6
Jig, plunger	1¼ to 0	1.63	4.3	78	0.088	6.0	97.3
Jig, basket	1 to 0	1.72	4.9	93	0.130	4.5	97.8
Wet table[d]	3/32 to 0	1.78	6.7	63	0.075	4.4	98.0
Wet table[d]	¼ to 0	1.70	1.9	79	0.103	2.5	99.3
Rheolaveur	4 to 5/16	1.63	10.4	75	0.096	8.5	98.8
Rheolaveur	5/16 to 0	1.70	13.5	130	0.294	7.1	94.4
Menzies cone	3 to 2	1.55	8.7	31	0.035	2.9	99.3
Menzies cone	¾ to ⅜	1.65	3.8	63	0.078	2.5	98.2
Hydrotator	11/16 to 28	1.83	4.2	63	0.099	2.6	99.0
Froth flotation	10 to 0	1.57	3.6	73	0.127	4.3	98.2
Froth flotation	10 to 0	1.69	4.7	125	0.234	7.2	96.4
Humphreys spiral[d]	8 to 0	2.04		147	0.367		91.8
Pneumatic table[d]	½ to ¼	1.68	2.0	94	0.131	3.2	98.3
Contraflow	2 to 0	1.79	1.8	98	0.179	3.0	99.2

[a] The specific gravity at which the distribution curve crosses the 50 pct ordinate, indicating equal distribution of material between washed coal and refuse.
[b] Not adjusted for material heavier than 2.0 sp gr.
[c] Fraser and Yancey formula: yield of washed coal divided by yield of float coal of some ash content.
[d] Laboratory test, all others plant tests.
[e] Cyclone.

erally results in a corresponding loss in efficiency. Examination of the distribution curves in both Figs. 9 and 10 demonstrates that with separations at higher densities the recovery of the lighter fractions of coal is improved, and this higher recovery is primarily responsible for the better efficiencies that are obtained in separations at higher densities. This condition applies to all but some heavy-medium operations.

The great influence that the recovery effected in the range of lower density has on efficiency directs attention to an important practical limitation in the use of error area as a criterion of washery performance. The error area representing the loss of the light coal is only a small part of the total, and yet this portion of the area exerts the most influence on efficiency. In one instance, for example, an increase in error area from 81 to 85 resulted in a loss in efficiency of over 6 pct because the increase in error area was entirely in the range of lower density. The same numerical increase in error area at either intermediate or high density would have decreased the efficiency only 1 pct. The common practice of expressing error area as a single figure has been followed in this report for the sake of simplicity, but the use of two separate areas representing the inclusion of impurity in the washed coal and the loss of coal in the refuse provides a measure of performance that can be correlated better with efficiency.

This limitation in the usefulness of error area should not be construed as presenting an advantage of probable error over error area, for probable error does not even take into account directly what recovery was effected in the lightest coal or what proportion of the heaviest impurity was rejected as refuse.

Table IV summarizes the results of 26 washing operations in terms of density of separation, efficiency, percentage of misplaced material, error area, and probable error. These data demonstrate clearly that there is no direct correlation between efficiency and misplaced material on the one hand and error area and probable error on the other. A sharp separation is an efficient one when made on a coal of favorable washability characteristics, but even a very sharp separation may result in a relatively low efficiency if made at a low density or if the coal contains an abnormal amount of material of intermediate density.

Summary

One of the principal objectives of the First International Conferences on Coal Preparation held in Paris in June, 1950, was to promote international standardization of the methods used for evaluating coal-washery performance.

Performance can be evaluated in a number of different ways, with the choice of the proper method to use being dictated by the objectives of the investigation and the data available. The performance of any cleaning unit is determined by the character of the raw coal treated, the density of the separation, and the sharpness of the separation effected between coal and impurity. Thus, three factors enter into performance; one is determined by the coal, one is dictated by market or use considerations, and one is an inherent characteristic of the cleaning unit itself.

Efficiency, which is influenced by all three factors, measures directly the loss of salable coal and thus is the performance criterion of greatest practical value.

Sharpness of separation is measured in terms of either error area or probable error, both of which are determined from a distribution curve for the washing operation. These criteria are useful in comparing the operation of washing units making dissimilar separations, because they are substantially independent of either the character of the raw coal or the density of separation. They must not be confused with efficiency, however, because low values of error area or probable error sometimes are accompanied by low efficiency.

Acknowledgments

Coal-preparation problems are investigated at the Northwest Experiment Station of Region II of the Bureau of Mines in cooperation with the School of Mineral Engineering of the University of Washington. The work is done under the general direction of Arno C. Fieldner, Chief, Fuels and Explosives Div., Bureau of Mines. The authors wish to acknowledge the assistance of Charles L. Allyn, Chemical Engineer, Bureau of Mines, in preparing the illustrations for this report.

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A Calorimetric Method for Studying Grinding in a Tumbling Medium

by A. Kenneth Schellinger

DURING the comminution of a brittle material in the presence of dry air, no known phase change or chemical reaction takes place. The energy changes associated with the comminution are those of the transformation of kinetic energy into heat energy and the energy absorption represented by the actual rupture of molecular and/or ionic bonds. The random nature of the actual grinding process in a tumbling-mill often precludes the effective application of a rupture force on each individual particle of brittle mineral present. The result must be a great deal of useless vibration on an atomic scale in both the brittle mineral and the grinding medium below the elastic limit, which vibration is eventually manifested as heat energy in the mill. The actual rupture of molecular, or ionic, bonds creates new surface on the brittle minerals; each square centimeter of this new surface then must possess a number of potential bonds, each of which could reform theoretically with the release of energy. The sum total of this potential energy of bonding per unit of new surface, minus bond satiation effects of the enclosing gas or liquid, must be the surface energy of the mineral. If the comminution of the brittle mineral were carried out within the chamber of a calorimeter, the energy released as heat could be determined rather accurately. Moreover, the total kinetic energy input to the comminuting mass (mineral and medium) conceivably could be derived by a suitable torquemeter and revolution counter if the whole calorimeter were revolved. The difference between these two energy figures should then represent the net energy (gross energy input—heat energy output) used in creating new surface, gliding or slip on atomic planes, twinning, and possibly other lattice rearrangements within the mineral. If it be postulated further that with a hard, brittle crystalline material, such as quartz, the energy used to produce gliding, twinning, and other permanent lattice movements is a very small fraction of the total net energy absorbed, then this energy absorption within the grinding calorimeter can be ascribed to the creation of new surface energy alone. Such energy is, from the thermodynamic standpoint, the true effective work of the grinding device, in this case a tumbling-medium mill. The ratio of this effective work (in calories) to the total energy input (in calories) should express the thermodynamic efficiency of the comminution taking place under the conditions used in the grinding calorimeter.

Table I. Calibration of the Grinding Calorimeter-Energy Balances without Brittle Charge

Run No.	Chamber Charge	Gram-calories Input	 Output	Diff.	Input, Pct
2	(dry) steel balls	440	430	—10	—2.3
3	(dry) balls	298	296	—2	—0.7
5	(dry) balls	407	408	+1	+0.2
14	Chilled C.I. shot (dry)	1260	1255	—5	—0.4
33	Chilled C.I. shot (wet)	1455	1455	0	0
38	Chilled C.I. shot (dry)	1375	1380	+5	+0.4
5A	Chilled C.I. shot (dry)	1410	1393	—17	—1.2
1A	Chilled C.I. shot (dry)	1405	1420	+15	+1.1
9SC	Chilled C.I. shot (dry)	1228	1220	—8	—0.6
11SA	Chilled C.I. shot (dry)	1358	1340	—18	—1.3
9SB	Chilled C.I. shot (dry)	964	963	—1	—0.1
		Input, avg pct:			—0.45

This approach to the measurement of the energetics of tumbling-mill grinding was suggested by some previous work by Fahrenwald and coworkers[1] with a laboratory ball mill and a thermometer. They measured the efficiency of grinding based on the formula:

$$\text{Pct Grinding Efficiency} = \frac{\text{Energy Input (calories)} - \text{Thermal Energy Output (calories)} \times 100}{\text{Energy input (calories)}}$$

Efficiencies thus measured, while grinding quartz sand, ranged from 19.7 pct to 7.4 pct. Cook in 1915,[2] used this approach to the problem of grinding efficiencies in stamp batteries and obtained the energy input by stamp weights times drop distances and the energy output by means of thermometers and pulp flow rates. He calculated that 80 pct of the input energy to the stamp shoes was wasted as sensible heat in the ore pulp. No other studies of comminution by this method of energy differences came to the author's attention.

Reasonable agreement as to the order-of-magnitude of comminution efficiencies in tumbling-media mills is lacking in the literature. There is a widespread impression among technical workers that the grinding efficiency of the industrial ball mill may be on the order of 1 pct, or even a fraction

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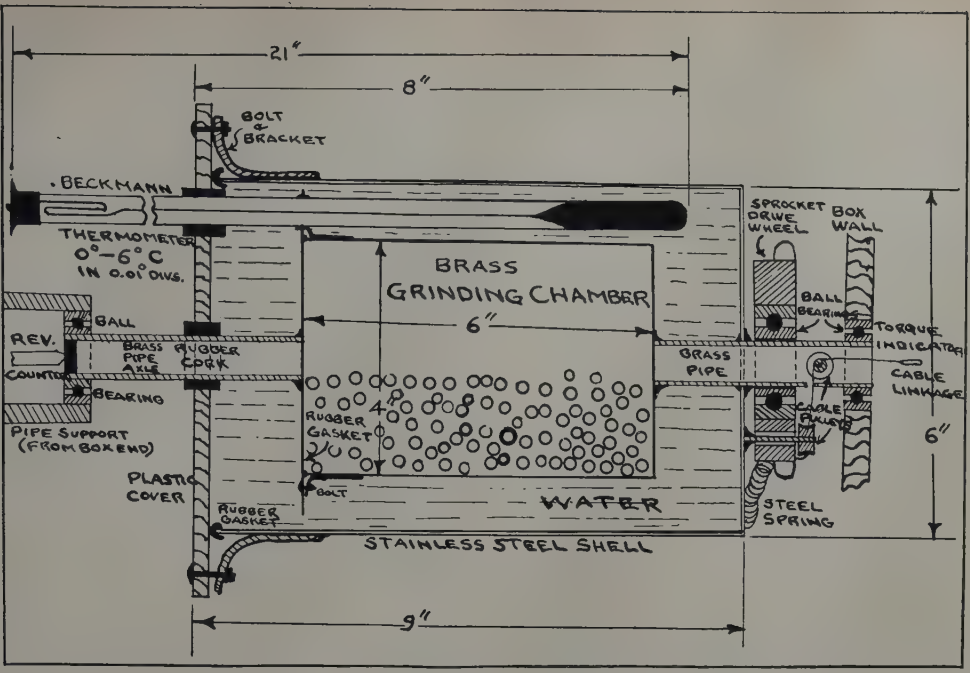

Fig. 1—Grinding calorimeter, longitudinal section.

of this. Values calculated from theoretical surface energy figures for quartz (Edser's value of 920 ergs per sq cm, and Martin's value of 520 ergs per sq cm) plus the surface areas of ground products have tended to be less than 1 pct. Johnson, Axelson, and Piret,[3] working on quartz, with drop-weight crushers and gas-adsorption surface area measurement have concluded that the efficiency of crushing is of this order of magnitude, 0.1 pct, if the theoretical surface energy figures are correct. However, Gross and Zimmerly[4] found the surface production efficiency of ore-grinding ball mills to be about 45 pct. Wilson[5] obtained similar efficiency figures ranging from 19 to 43 pct for ball mills grinding cement clinker. Bond and Maxson[6] found efficiencies in the neighborhood of 60 pct during the grinding of various ores in connection with their standard grindability test. All of these workers avoided acceptance of theoretical surface energy values by use of the drop-weight crusher, or a similar device, as a standard of comparison. In effect, then, surface energy figures obtained by the use of the drop-weight crusher plus several methods of estimating the new surface produced, have been used to show that the ball mill is from 20 to 60 pct efficient as a surface producer. Theoretical surface energy figures for quartz, on the other hand, have condemned the commercial ball mill to the unenviable category of better than 99 pct efficiency as a heat energy producer.

Calorimeter Construction

Construction of a calorimeter to embody the principles and objectives outlined meant that a suitable grinding chamber had to be enclosed completely in the calorimetric fluid—water in this case. To impart a tumbling action to the contained grinding medium, the chamber was designed as a 6x4 in. cylinder turning on its long axis in a horizontal position. This was the tumbling-mill portion of the calorimeter. The calorimeter is shown in Fig. 1. The energy output from the grinding chamber was readily measurable with this calorimetric assembly to a good degree of accuracy.

However, to derive the input energy with an equal degree of accuracy meant the construction of a suitable torquemeter that could record the net turning torque exerted on the tumbling medium at all times. Frictional torque was essentially zero, inasmuch as the calorimetric assembly, when filled with water but with no grinding medium (shot), could be spun easily on its ball bearings. Experimentation with dc motor efficiencies and other methods, convinced the author that sufficient accuracy could not be obtained in this direction. The device finally decided upon was modified from one described by Gross and Zimmerly[7] to determine work input to a laboratory ball mill. The modified device consisted of a sprocket wheel fitted to the calorimeter closed-end axis, shown in Fig. 1, by means of a ball-bearing race, so that it revolved freely in relation to the axis. Driving torque was transmitted to the calorimeter proper by means of a steel spring fastened tangentially between this sprocket wheel and the bottom of the water jacket. Increased turning torque meant increased spring extension, which was transmitted to an outside recorder by means of a taut cable system. This recording torquemeter system is shown in Fig. 2.

The entire assembly was enclosed in a wooden box with a glass cover to permit control of thermal losses from the water jacket shell. Some details of this enclosure are shown in Fig. 2. Calorimetric thermometers were inserted through the top of the box so that the air temperature on each side of the

Table II. Preliminary Load-Efficiency Data for Dry and Wet Grinding in the Grinding Calorimeter

Charge	100		200		300		400	
Quartz, g	Dry	Wet	Dry	Wet	Dry	Wet	Dry	Wet
Energy, grinding calorimeter								
Input	1207	1185	1100	1248	1345	1270	1215	1230
Output	1092	1020	927	1050	1090	1025	1080	1130
Net (to new surface)	115	165	173	198	255	245	135	100
Efficiency, pct	9.7	13.9	15.7	15.9	19.0	19.3	11.1	8.1
Product								
—325-mesh, g	14.7	39.6	32.6	33.4	46.5	39.3	42.8	44.4
Charge, pct	14.8	39.6	16.3	16.7	15.5	13.1	10.7	11.1

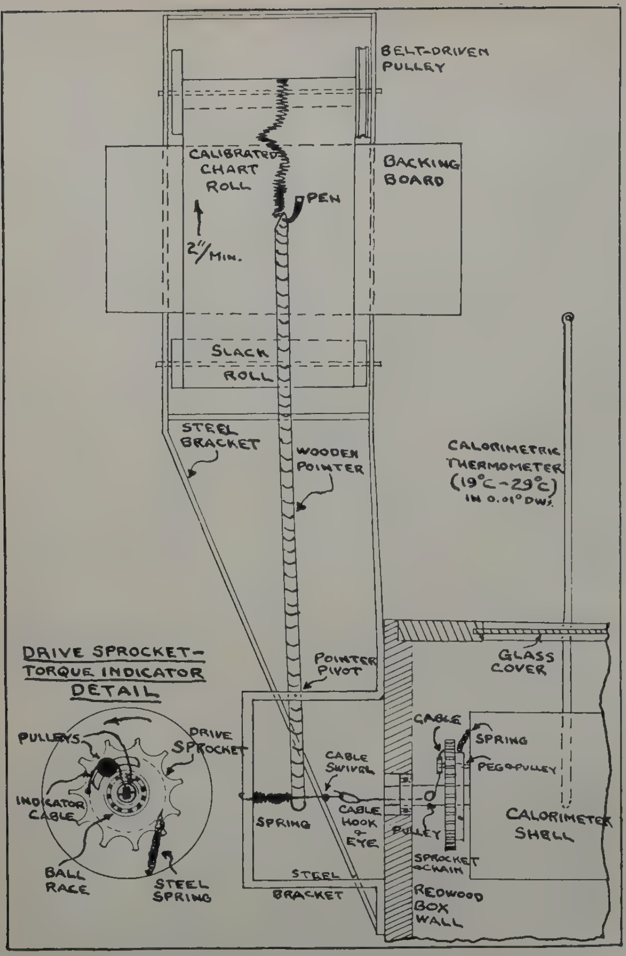

Fig. 2—Recording torquemeter assembly.

calorimeter shell was known. These thermometers were read to 0.01°C. A small fan driven by a stirrer motor was inserted in each side of the box to keep the air within thoroughly mixed. A sensitive thermo-regulator also was hung within the box beside one of the calorimetric thermometers. This controlled a solenoid air valve which, in turn, supplied the box with cold compressed air from a copper coil submerged in an ice bath. By this means the air temperature around the calorimeter could be held at any desired figure between 25° and 15°C, to within 0.1°C by adjusting the thermoregulator. Thus thermal potential between the calorimeter water and the outside air could be set to any convenient figure. Curves were then drawn up by plotting thermal leakage (degrees per minute) against thermal potential; one curve was for the calorimeter shell while stationary and another while revolving at about 70 rpm. The completed box-calorimeter assembly required careful calibration before use in actual grinding studies.

Instrument Calibration

Calibration of the assembled instrument was accomplished by evaluation of the following:

1—The calorimeter constant in terms of calories per hundredth of a degree centrigrade temperature rise on the Beckmann thermometer.

2—The value of the scale on the recording torquemeter chart in terms of foot-pounds of energy per revolution of the calorimeter.

In addition, balances of input energy against output energy were made when no useful work was done in the grinding chamber, that is, when no brittle material was present. These are reported in Table I.

The calorimeter constant, 29.6, was derived by releasing a known amount of heat within the grinding chamber under conditions of temperature equilibrium and noting the resultant temperature rise on the Beckmann thermometer. Torque values in foot-pounds per revolution were translated to recording torquemeter scale values by hoisting a known weight through a definite distance and allowing the resulting pointer deflections to mark on the torquemeter chart.

Preliminary Experimental Results

The grinding calorimeter comminutes brittle solids in much the same manner as does a large industrial ball mill; that is, by enveloping small particles of the solid in a tumbling mass of larger spherical grinding medium where compressive and frictional stresses tend to rupture the brittle material. Therefore it was thought that an investigation of the absolute efficiency of surface production in the calorimeter might be more directly comparable to large-scale practice than would an investigation by means of drop-weight crushers or other devices. Control of many but not all of the variables found in large-scale tumbling-mills is possible in the calorimeter. These include relative weights of charge and grinding medium, shape and size distribution of grinding medium, speed of rotation, grinding fluid, pulp density, and others. Major disadvantages in a direct comparison with large-scale grinding are the impossibility of reproducing the particle-ball contact intensity factor and the necessity of using a batch grinding technique.

Loading and the Thermodynamic Efficiency: Preliminary experimental work was concerned with discovering the most efficient loading of the mill with brittle material, for it was found early in the work that the loading factor had a large influence on the grinding efficiency. Technique consisted of varying the weight of brittle charge, while holding the other factors such as speed, kind of grinding medium, weight of grinding medium, etc., constant. For each weight of charge the difference between input and output energy was determined by a standard experimental procedure and the grinding efficiency was computed by the relation:

$$\text{Pct thermodynamic grinding efficiency} = \frac{\begin{matrix}\text{(Input)} & - & \text{(Output)} \\ \text{(energy)} & & \text{(energy)}\end{matrix} \times 100}{\text{(input energy)}}$$

This was done with a dry charge over a range of weight values, and then over the same range with the charge mixed with water in the ratio 60 charge: 40 water, or 60 pct solids. Hence these two sets of experimental values on the same brittle mineral formed a comparison of the energy efficiencies of dry and wet grinding under otherwise similar conditions.

The brittle mineral used was a glass sand composed almost entirely of rounded quartz grains and was screened to give grains ranging in size between 100 and 200-mesh. That is, the approximate average grain diameter of the quartz charge was 0.0044 in. The average diameter of the chilled shot charge was approximately 0.185 in., which would give a ball: charge diam ratio of about 42. This might correspond to a ball mill using a 3-in. grinding ball to

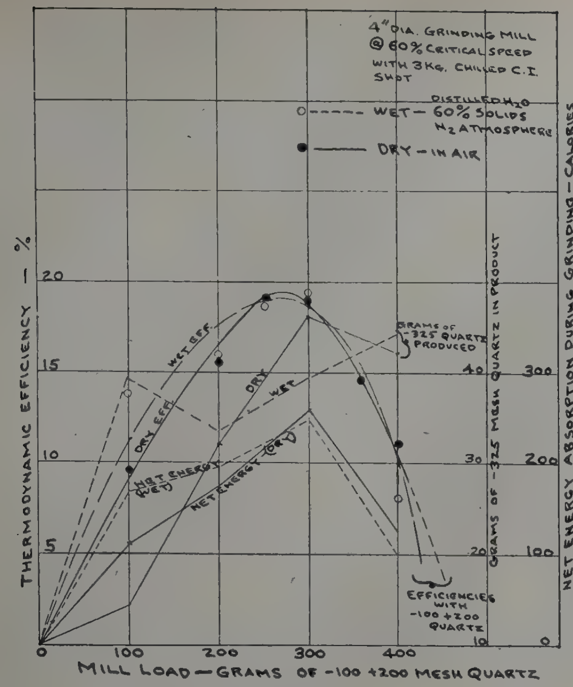

Fig. 3—Mill loading vs. efficiency and —325-mesh in product for wet and dry grinding in the grinding calorimeter.

grind approximately 10-mesh mineral. Further, the mill: ball diam ratio for the calorimeter was 21.6, which corresponds to a 5 ft 4 in. diam ball mill using 3 in. balls as a grinding medium—admittedly a rather improbable commercial situation. The length of the grinding time for each determination was about 30 min or 2100 rev at 70 rpm, which roughly corresponds to 60 pct of the apparent critical speed of the mill as shown by the torquemeter.

Wet runs were made by the use of distilled water and tank nitrogen gas to avoid heat-producing oxidation effects. For these the grinding chamber was pumped down by means of a vacuum pump after charging with quartz and water, and the air was replaced with nitrogen. Several repetitions of this treatment gave an atmosphere within the grinding chamber that produced no noticeable oxidation of the ferrous grinding medium. A correction was made in the calorimetric constant to allow for the water in the grinding chamber.

After each run the grinding chamber was opened and the quartz charge screened on a —325-mesh Tyler sieve to provide an approximate indication of the amount of surface produced. The charge had no —325-mesh material prior to grinding. Wet ground material was washed from the grinding chamber and shot medium by means of alcohol to inhibit rusting of the shot. Data collected during these dry and wet efficiency runs are presented in Table II and also in Fig. 3 where the plotted efficiency data trends are averaged by means of smooth parabolic curves.

Fig. 3 indicates that the thermodynamic efficiencies during both wet and dry grinding of quartz sand in the calorimeter range from 10 pct to 19 pct and are dependent on the loading of the grinding chamber. The most efficient loading of the grinding chamber appears to be near 300 g, with efficiencies dropping off on each side of this peak in a parabolic fashion. The efficiency-load curves for both wet and dry grinding show no significant differences, which is somewhat unexpected inasmuch as wet grinding is generally regarded as more efficient than dry grinding. It must be remembered, however, that the plotted thermodynamic efficiencies are for surface production rather than for a certain weight per kw-hr finer than a chosen mesh size. The total grams of —325-mesh quartz produced in each case is a very rough measure of the new surface production and is seen to follow the efficiency curves approximately when plotted on Fig. 3.

Pulp Density and Thermodynamic Grinding Efficiency: A number of runs were made with a constant solids charge of 250 g of quartz sand ranging in size between 100 and 200-mesh. The variable employed was the degree of wetness of the charge, that is, the quantity of water was varied from 800 to 0 g corresponding to pulp densities of 30, 40, 60, 80, and 100 pct solids. Corrections then were applied to the calorimeter constant to allow for the heat capacities of these varying amounts of water in the charge. All other variables were held constant except the charge level in the grinding chamber which rose with dilution.

Results of these runs are shown in Fig. 4. Efficiency is seen to have a downward trend towards the pure water end as might be expected. This trend is small in the practical grinding range, for efficiencies drop only from 18.5 pct at 100 pct solids (dry grinding) to 14.5 pct at 30 pct solids. Such a variation is probably within the experimental error inherent in this first model of the calorimeter and is contradicted somewhat by the comparison shown in Fig. 3.

Summary and Conclusions

1—A calorimetric technique has been developed for investigating the small-scale comminution of brittle minerals by the tumbling-mill principle.

(a) Construction of a revolving grinding calorimeter is described wherein a brittle mineral is ground by tumbling cast-iron shot in a 4-in. diam brass grinding chamber totally enclosed in a water jacket.

(b) Calibration and use of the calorimeter is described. When properly calibrated, the instrument will give a balance between input energy and output energy without brittle mineral in the grinding chamber.

2—Preliminary experimental runs with the grinding calorimeter on —100 +200-mesh quartz sand have yielded the following results:

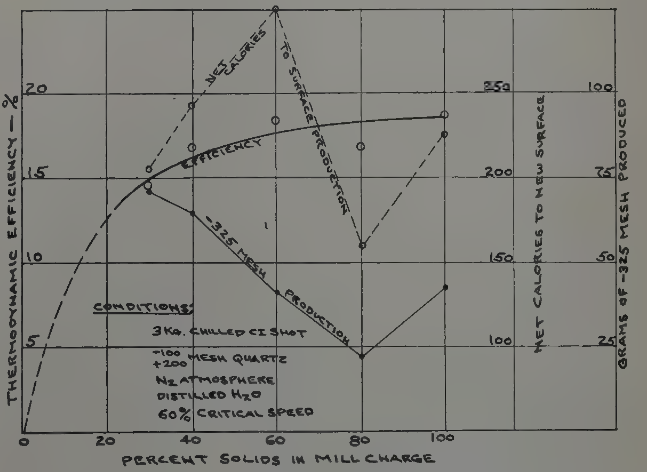

Fig. 4—Pulp density vs. efficiency and —325-mesh in product.

(a) That the absolute thermodynamic efficiency of surface production in the grinding chamber is a function of mineral quantity, of pulp density, and possibly of a number of other variables.

(b) That during dry grinding and during wet grinding at 60 pct solids, the thermodynamic efficiencies shown by the instrument varied from 10 to 19 pct, with the higher figure occurring at an optimum loading of 270 g of quartz sand.

(c) That there was no significant difference in peak efficiency between dry and wet grinding under the given conditions, i.e., 60 pct solids, 60 pct critical speed, —100 +200-mesh quartz.

(d) That the efficiency curve dropped slightly with increasing dilution of the charge with water, i.e., from 19 pct at 100 pct solids to 14.5 pct at 30 pct solids.

3—The establishment of an order-of-magnitude calorimetric value for the thermodynamic efficiency of the tumbling-mill type of grinding is regarded as one of the most significant results of the research. The optimum value found, 19 pct, agrees qualitatively with values found by earlier investigators using thermal methods on laboratory mills and is about 50 pct of the value found by Gross and Zimmerly for commercial ore-grinding ball mills.

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Technical Note

Beaker Flotation as a Quantitative Tool in Flotation Testing

by William Loblowitz

A simplified system of flotation testing has been developed recently at the Bureau of Mineral Research of Rutgers University. The technique involves the known principle of skin flotation and can be used with minute quantities of sample and reagents in a beaker. A vessel containing the conditioned sample with added water and flotation reagents is held inclined and rotated slowly. Water-repellent particles come in contact with air and float on the surface of the liquid, while wettable particles are submerged. The float is poured off and filtered whereas the nonfloat remains in the beaker. The filtrate may be returned to repeat the process.

This procedure can be made quantitative. Separations are obtained and are remarkably complete and repeatable. The effect of different reagents, optimum quantities, pH and other variables can be determined in a fraction of the time required for conventional testing. However, the method gives no information on the effect of frothers and the quality of froth to be expected in a cell, and it is not practical for particles finer than 200-mesh. Unlike other preliminary tests, separate float and nonfloat products are obtained, and their contents can be determined quantitatively by chemical analysis or grain counts. After the optimum reagent combination has been determined, the effect of fines and frothing may be verified in the conventional flotation cell. On the other hand, agglomeration-tabling can be directly duplicated from beaker tests.

Procedure: Five to ten grams of sample are weighed into a 100-ml beaker. Flotation reagents and a small initial quantity of water are stirred together; additional water is then added to produce the proper water-solids ratio. The beaker is then tilted to 45° and rotated slowly. After three to five rotations, liquid and float are poured off and filtered; the filtrate may be recycled until the separation is completed.

One test generally requires less than 10 min. Reagent concentration and pH remain quite constant because of the recycling of the filtrate. Two to four tests may be made simultaneously, and the effect of varied conditions may be observed by comparison. Considerable time is saved in preparation and handling since small quantities of sample suffice for a series of systematic tests.

Repeatability

One test was repeated four times under constant conditions. The sample used was a quartz sand containing approximately 24 pct feldspar and some clay aggregates. The +28-mesh grains had been screened out, and most of those finer than 100-mesh had been removed by hydraulic classification. The results are presented in Table I.

For comparison, 500 g of the sample were separated by agglomeration-tabling using the same reagent combination and quantities as given in Table I. The weight percent float was 18.3 pct, composed almost entirely of feldspar and with only traces of quartz. Most of the remaining feldspar was recovered in a middling product with 2.2 pct by weight.

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Table I. Flotation Test on Quartz Sample

Test Number	1	2	3	4	Avg	Avg Deviation
pH at start	3.15	3.25	3.20	3.15	3.19	0.04
pH at end	3.20	3.28	3.25	3.22	3.24	0.03
Weight pct float	20.65	20.25	21.20	21.15	20.81	0.38
Quartz in nonfloat,[a] pct	96.8	98.0	97.4	97.2	97.4	0.35

[a] (Grain count: only +80-mesh grains counted)

Reagents:	Hydrochloric Acid	3.3 lb per ton
	Sodium Fluoride	1.1 lb per ton
	Armac CD	0.17 lb per ton
	Fuel Oil	5.5 lb per ton

Float removed by pouring six times.

Float products contained over 96 pct feldspar, some clay aggregates and few fine quartz grains.

The nonfloat contained over 98.5 pct quartz as determined by grain count. This separation agrees well with the results obtained by beaker-flotation.

Other tests were made on the classifier overflow product of the same quartz sand that contained 90 pct —100-mesh and 23 pct —200-mesh. No effective separation was obtained. After screening out the —200-mesh particles, however, separations of quartz and feldspar equivalent to those in a flotation cell were obtained. For tests in the range of 100 to 200-mesh, 5 g samples were used, and the decanting of the float had to be repeated 16 times to achieve complete separation.

Mechanical Rotation

To reduce the human factor in rotating the beaker, it was found convenient to use a laboratory motor turned upside down so that its shaft points upward at an inclination of approximately 45°. Instead of the shaft, a short bolt is inserted into the chuck. A clamp grips the 100-ml beaker, which may be rotated at the desired inclination by running the motor at low speed. The motor is mounted on a laboratory support above a filter funnel, so that the beaker may be emptied into the filter by tilting the motor. Motor and beaker are then returned to the 45° position and the filtrate is returned to the beaker to repeat the separation cycle. This apparatus is not essential for the application of the method.

Field Testing

By using this method a rapid flotation test may be performed in the field far from the laboratory. Equipment would include a small mortar and pestle, two 2-in. diam sieves, beaker, funnel, filter paper, and reagent solutions in dropper bottles.

Summary and Conclusions

1—A rapid quantitative method for testing minerals by "skin" flotation was established.

2—Only 5 to 10 g of sample and no complicated equipment are required.

3—Particle size limits are 10 to 200-mesh.

4—Tests were reproduced with an accuracy comparable to careful testing with a laboratory flotation cell; the average deviation was 0.38 pct for four parallel tests.

5—The method may be used in the field with simple equipment.

6—A device for mechanical rotation and tilting is described but is not essential to the test.

Acknowledgment

Thanks are due to J. H. C. Martens for his friendly cooperation in performing the grain counts and to Helgi Johnson for editing this paper.

Technical Note

Approximation of the Energy Efficiencies of Commercial Ball Mills by the Energy Balance Method

by A. Kenneth Schellinger and Rustam D. Lalkaka

If the ball mill is considered only from an energy standpoint, it can be thought of as a converter of kinetic energy into heat energy and surface energy. The law of the conservation of energy must apply to the ball mill as it does to any machine or process in nature. The heat energy output of the ball mill is apparent to any casual observer of the operating mill since the outflow pulp will be appreciably warmer than the inflow water and rock. The rock in the ball mill is crushed by shearing of its ionic bonding caused by some impact-induced stresses beyond the elastic limit. As the bonds holding the mineral crystal lattices together are ruptured, new surface area is created on each unit of which a definite number of unsaturated bonds exist. Such unsaturated energy of bonding may be termed the "surface energy," and is a potential energy figure. This created energy was withdrawn from the ball mill system and represents an absorption, or disappearance of heat. To reduce the size of the rock, surface area must be created and with it the attendant characteristic surface energies of the minerals concerned. Thus the creation of surface energy, from an energy viewpoint, is the creation of useful work by the ball mill.

Because of the controversial nature of the energy efficiency figures for modern operating ball mills, it seemed worthwhile to approximate the ordinary magnitude of such efficiencies from the above viewpoint. This was done by an energy balance of the three aforementioned forms of energy, kinetic, heat, and surface energy, for two 9½x10-ft grate-type ball mills grinding cement raw materials. Kinetic energy input to the balls was approximated as closely as possible while the heat energy output was measured by means of a thermometer and water and rock flow rates. The assumption was made

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Table I. Data for Energy Balance Approximations

	Primary No. 1 Run 1	Primary No. 1 Run 2	Primary No. 2 Run 1
Feed, tons per hr	64	70	60
Outflow, pct solids	68.5	69.5	56.0
Power input, hp	480	526	445
Temperatures, °C			
Rock input	21.8	22.3	22.2
Water input	24.0	27.8	27.8
Pulp outflow	28.9	31.9	30.0

that the low noise level represented a negligible amount of energy as did any phase changes, oxidation effects, etc. in the rock mass. These considerations led to the energy efficiency approximation formula:

$$\text{Pct energy efficiency} = \frac{\begin{array}{c}\text{kinetic} \\ \text{energy input} \\ \text{(calories)}\end{array} - \begin{array}{c}\text{thermal} \\ \text{energy output} \\ \text{(calories)}\end{array}}{\text{kinetic energy input (calories)}} \times 100$$

Measurement of energy efficiencies of a small tumbling-medium mill has been made by Fahrenwald[1] and coworkers. These efficiencies, during the grinding of quartz, varied between 7 and 19 pct. A grinding calorimeter, using the tumbling-medium principle, has been used by one of the authors[2] to obtain energy efficiency figures in the range 10 to 19 pct while grinding quartz sand. Application of the method to large operating grinding mills has not been reported in the literature, although Cook[3] long ago applied it to an operating stamp battery and showed that over 80 pct of the input energy to the stamp shoes was degraded as heat energy.

Method and Data: Energy approximations for the purpose of computing balances on operating mills required the use of a sensitive thermometer to ascertain the temperatures of the various streams of rock, water, and pulp flowing into and from the mills. Flow rates were obtained from known operating data on the mills or were measured, if necessary. Temperature of the rock feed stream was taken by repeatedly immersing the sheathed bulb of the thermometer in the moving rock on the belt and reading the equilibrium temperatures obtained. The percentage of solids in the pulp stream leaving the mill was determined by the usual method of weighing a fixed volume of the pulp and reading the percentage of solids from a prepared chart. From this determination the tonnage of water entering the mill with the rock could be calculated. Temperature of the water was taken by immersing the thermometer bulb in the water jet feeding into the mill feed-scoop box. The temperature of the pulp stream leaving the mill for the classifier was taken in a similar manner. Final temperature figures in each case were obtained by averaging a number of readings, each taken at a different point in the material stream concerned. Data for energy balance approximations are given in Table I.

Table II. Energy Balances

	Primary No. 1 Run 1	Primary No. 1 Run 2	Primary No. 2 Run 1
Energy inputs, 10^6 g cal per hr	237.5	259.5	219.5
Heat outputs, 10^6 g cal per hr	190.8	233.0	175.9
Surface energy, 10^6 g cal per hr	46.7	26.5	43.6
Efficiency, pct	19.7	10.2	19.9

The motor power input at the time of the temperature reading was taken from a polyphase watthour meter and was checked against the average power drawn during the previous 24-hr period. Variations in the power draft of the mills were quite small.

The designations and types of ball mills on which these measurements were made are as follows:

Primary No. 1	Allis-Chalmers 9½x10-ft grate-discharge ball mill, 500-hp drive motor
Primary No. 2	F. L. Smidth 9x11½-ft grate-discharge ball mill, 500-hp drive motor

The average specific heat of the rock input to the mills (approximately $CaCO_3$, 84 pct; SiO_2, 16 pct) can be calculated from well-known equations as 0.1935 at 27°C. The specific heat of water at the same temperature is 0.9976. The net energy input to the balls alone is obtained by applying a factor (0.77) to the energy input to the motor. This factor was obtained from the literature where it was found that Gow and Guggenheim[4] had packed a normal ball load around the axis of a 6x4-ft mill and had run it at 80 pct of critical speed without any ball motion in order to obtain the power loss caused by friction alone. In the absence of any direct data on larger mills this factor was used for the energy balance approximations.

Radiation and convection loss or gain by the rotating mill shells is a distinct possibility if the surrounding air temperatures differ from that of the mill shells themselves. The radiation and convection rate depends upon a number of factors such as the surface finish and shape of the shell, the temperature of the shell, and the thermal gradient between shell and air. These are complex factors to evaluate and no attempt was made to do so in these approximations when it was found that the air temperature in the neighborhood of the mill shells was substantially the same as the average temperature in the mill.

The results of calculations made by and with the foregoing methods and data are shown in Table II.

The thermodynamic efficiencies obtained by the foregoing approximations appear to confirm the few results obtained by other workers who have used this approach. Hence, Fahrenwald's[1] efficiencies, 7 to 19 pct, on a laboratory ball mill are very similar to the mill efficiencies, 19.7, 10.2 and 19.9 pct shown in Table II. Results obtained with the grinding[2] calorimeter (10 to 19 pct) are also in the same range as reported for these primary mills.

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Economic Aspects of Ground Water in Florida

by V. T. Stringfield and H. H. Cooper, Jr.

ONE of the earliest investigations of ground water in Florida was made in 1513 when Ponce de Leon arrived at St. Augustine in search of the Fountain of Youth. The history of the development of the water resources of the State shows that the large artesian reservoir that underlies Florida was discovered in the latter part of the last century. Part of that history is given by L. C. Johnson[1] who states that the first successful artesian well in Florida was drilled at St. Augustine between 1880 and 1882. After the City of Jacksonville failed to obtain a flow of artesian water at a depth of nearly 400 ft and abandoned the drilling, R. N. Ellis, City Engineer, and L. C. Johnson, using Johnson's knowledge of the geologic structure of the artesian reservoir, estimated correctly that artesian water could be obtained at a depth of about 500 ft. Thus began the development of the large artesian system in the northeastern part of the State. In 1908, the Florida Geological Survey issued its first report on the ground water of central Florida.

More recent reports give the results of investigations that have been in progress in cooperation between the Florida Geological Survey and the U. S. Geological Survey since 1930. As a result, the ground-water geology and hydrology of Florida are now so well known that ground-water problems such as confronted early investigators no longer exist. However, new problems arise and new discoveries are made as the demand for more ground water increases with the development of the State.

Water-Bearing Formations

Descriptions of the geologic formations and a map showing their distribution at the surface are given in a recent report by Cooke.[2]

The geologic formations that yield the ground-water supplies in Florida represent only a small part (about 1000 ft) of the total thickness of the sedimentary rocks (more than 15,000 ft) that underlie Florida. The water-bearing rocks that yield fresh water include more than two dozen formations that range in age from Eocene to recent.

The Eocene formations, which consist chiefly of limestone, are the oldest and the deepest of the formations in Florida that yield fresh water to wells. In 1944, on the basis of a study of the foraminifera, the Applins[3] divided the limestone of Eocene age into six formations, which are as follows, from top to bottom:

Age or group	*Formation*
Jackson	Ocala limestone
Claiborne	Avon Park limestone
	Tallahassee limestone
	Lake City limestone
Wilcox	Salt Mountain limestone
	Oldsmar limestone

The Ocala limestone and the underlying formations of Claiborne age, along with some of the overlying limestone of Oligocene and Miocene age, constitute the principal source of water in Florida and southeastern Georgia and generally may be regarded as forming one artesian aquifer or water-bearing unit. This aquifer is referred to in this paper as the Floridan aquifer, a name proposed by Parker.[4]

In part of Seminole County alone more than 1000 flowing wells yield water from the Floridan aquifer. A yield of 6500 gpm, or about 9.5 million gal per day, by natural flow from a well penetrating the aquifer at Jacksonville, Florida, was observed in 1942. A yield of about 7000 gpm by natural flow was reported for a well 1390 ft deep at St. Augustine in 1887. The largest yield reported from a pumped well penetrating the aquifer is 7500 gpm, or about 10.5 million gal per day. The aquifer is also the source of some of the large springs, such as Silver Springs, whose discharge, according to measurements made by the U. S. Geological Survey, has ranged from 526 to 1350 sec-ft, or from 340 to 872 million gal per day.

The top of the Ocala limestone, as represented by contours in Fig. 1, indicates in a general way the geologic structure of the formations that comprise the Floridan aquifer. The Ocala is at or near the surface in the areas represented by shading in the northwestern part of the peninsula and also in an area in western Florida adjacent to Alabama and

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Georgia. In areas where it is structurally high, as in the central part of the State, it is cavernous. Sinkholes extending from the surface of the ground into the limestone are numerous where the limestone is at or near the surface.

Of the formations of Oligocene and Miocene age, the Tampa limestone of Miocene age is one of the most important as a source of artesian water. The Tampa overlies the Ocala limestone in the peninsula except locally where the Suwannee limestone[6-8] of Oligocene age (formerly known as the base of the Tampa) is present between the Tampa and Ocala limestones. The Tampa is at or near the surface in the west-central and northwestern parts of the peninsula, and in some of the area west of the Suwannee River. Well records indicate that it is not present in the eastern and northeastern parts of the State. The Tampa is the source of some of the large springs, such as Wakulla Springs, near Tallahassee.

The Hawthorn formation of Miocene age, or its equivalent, is present throughout the State, except where it has been eroded away and the older rocks are exposed. It overlies the Floridan aquifer and is in turn overlain by younger materials.

The Hawthorn consists of about 400 ft of interbedded marl, limestone, and clay, most of which is more or less phosphatic. In western Florida, west of the Apalachicola River, the possible equivalent of the Hawthorn is mapped and described[9] as the Chipola and Shoal River formations.

The Hawthorn formation contains permeable water-bearing beds in which water is confined by less permeable beds that lie above and below. The Hawthorn as a whole prevents the upward escape of much of the artesian water in the underlying limestones of Miocene, Oligocene, and Eocene age.

The surficial sands and the rocks that overlie the Hawthorn formation supply water to many shallow wells for domestic supply and also to some of the wells for public supplies—as those for Miami, Pensacola, and other cities in coastal areas where the deeper water is highly mineralized. Some of the shallow formations yield large supplies of nonartesian water. For example, the Tamiami formation of Pliocene age is one of the most productive formations ever investigated by the Geological Survey. This formation is the source of the municipal water supply of Miami, Miami Beach, and Coral Gables.

Source and Movement of the Water

The movement of the water in the artesian limestones in Florida is much more widespread than the movement of the water in the overlying formations. Water that enters the artesian limestones at some places travels more than 50 miles before it reaches an exit through a well or a spring. This fact is generally recognized and perhaps is responsible in part for the false notion that the artesian water of Florida comes from very remote places such as the Appalachian Mountains, the Great Lakes,[10] or the Rocky Mountains. All who are familiar with the groundwater geology of Florida recognize that the chief source of ground water is the rain that falls within the State and the water that moves from the southern parts of the adjacent States of Georgia and Alabama. However, much remains for the investigators of ground water in Florida to learn concerning the source and movement of the artesian water.

A report[11] of the Florida Geological Survey prepared in 1929 includes a map that represents the pressure head of the artesian water in the Ocala limestone in a large part of Florida. In 1933, the U. S. Corps of Engineers prepared a map showing contours representing the pressure head of the artesian water in north-central and northern Florida.

Confronted with the need for a better understanding of the occurrence, source, and movement of the artesian water, the U. S. Geological Survey, in cooperation with the Florida Geological Survey, in 1933 and 1934, prepared a map[12-14] of the Florida peninsula showing contours of the piezometric surface—an imaginary surface representing the height to which water will rise with reference to sea level in tightly cased wells that penetrate the Floridan aquifer. In the following year the two Surveys prepared a map of the piezometric surface[15] in the remainder of the State. These two maps proved to be so valuable in revealing information on the artesian water that they encouraged, in 1938, the mapping[16] of the piezometric surface in southeastern Georgia by the U. S. Geological Survey, in cooperation with the Georgia Department of Mines, Mining, and Geology. The piezometric contours in Florida and Georgia are given as Fig. 2. This map apparently covers the largest area for which a piezometric map has yet been prepared and is remarkable in that it shows the areas of recharge and discharge and the direction of movement of the artesian water over hundreds of miles. It also reveals important areas of recharge that were unknown before its preparation. The piezometric surface in the Florida peninsula, as shown in Fig. 3, has been described in a report by Stringfield,[17] and that in Georgia has been described by Warren.[18]

One of the most conspicuous features of the piezometric surface is in the central part of the Florida peninsula, in Polk and adjacent counties, where the surface has the shape of an elongated dome, part of which is 90 to 120 ft above sea level. This dome indicates local recharge of the water-bearing formation. Thus, recharge occurs south of the large area in which the Ocala and older limestones are exposed at the surface (see shaded area in Fig. 1). In the Polk County area the Ocala and Tampa limestones are overlain by relatively impervious beds of the Hawthorn formation, and therefore it might appear that little or no recharge would occur there. However, within the area are many lakes occupying old sinkholes now partly filled with permeable sands that permit downward percolation of water into the underlying limestone. As indicated by the absence of surface streams, the rainfall drains into the numerous lakes and depressions, providing a source of water for recharge of the formations. Water from the lake region of central Florida, therefore, supplies most of the artesian water in central and southern Florida.

Another area in which the piezometric surface is high includes the northwestern part of Putnam County and adjacent parts of Clay, Bradford, and Alachua Counties. In this area the piezometric surface ranges from about 50 ft to as much as 90 ft above sea level. Northward, in Bradford and Clay Counties, it forms a ridge 70 to 80 ft above sea level and extends at that altitude into Georgia. Recharge occurs in this area through sinkholes, or directly into the formations where they are exposed at the surface in the southern and southwestern parts of the area. In the northern part of the area, although the piezometric surface is as much as 70 ft above sea level, little, if any, recharge takes place locally, but

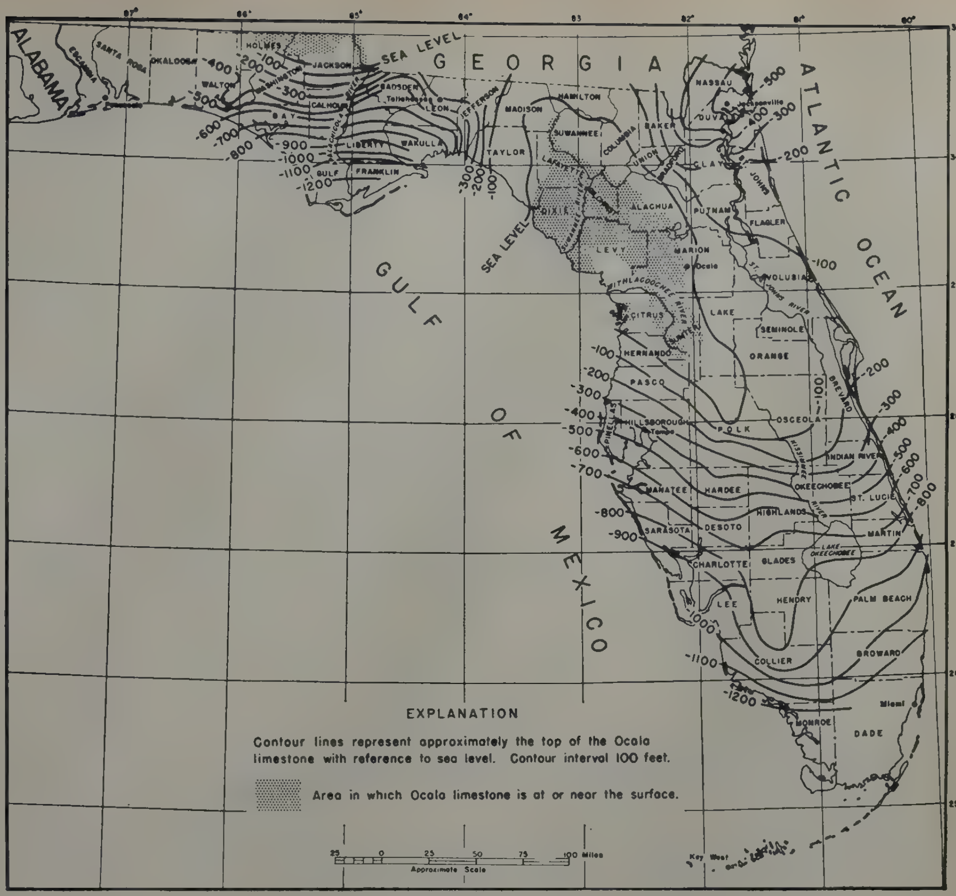

Fig. 1—The top of the Ocala limestone in Florida.

water is supplied through a recharge area in Georgia. Other areas in which the piezometric surface is relatively high, as where it forms a ridge extending from southeast Putnam County into Volusia County and where it forms a dome north of Tampa in Pasco County, indicate recharge areas.

Recharge occurs also in Marion County, where the piezometric surface forms a broad saddle between the Polk County and Putnam County areas. However, large quantities of ground water discharge in this area through large springs, such as Silver Springs and Rainbow Springs. Large areas of discharge occur where the piezometric surface is low along parts of the east and west coasts because in these areas the Ocala limestone is at or near the surface. There is also a large discharge area in the valley of the St. Johns River. Along the valleys of some of the rivers, such as the Withlacoochee and the Suwannee, where the water-bearing formations are exposed, the ground-water levels rise and fall with the stage of the river.

In Florida west of the Suwannee River, as might be expected, the general direction of the movement of the ground water is toward the Gulf Coast. One prominent feature of the piezometric surface in western Florida is the large embayment that follows the outline of the Gulf Coast in Wakulla, Taylor, Leon, and Jefferson Counties. In much of this area the Tampa limestone is at or near the surface and receives recharge directly from the rainfall. However, the piezometric surface is low because the rock is very permeable and cavernous and permits the artesian water to move relatively freely to the Gulf. East of the embayment, in Madison, Taylor, Lafayette, and Dixie Counties, the piezometric surface has the shape of a ridge extending, in general, in a southward direction, indicating that recharge occurs locally. The well data and the shape of the piezometric surface indicate that in this area the rocks are generally less permeable than in the area to the west.

In western Florida the piezometric surface is highest in Holmes and Jackson Counties, where the water-bearing formations are at or near the surface of the ground. In Escambia and Santa Rosa Counties the Tampa and Ocala limestones, or their equivalents, are far below the surface and are penetrated by only a few wells; therefore, the data are not sufficient for mapping the piezometric surface of these formations. However, the piezometric surface of the artesian water in the younger formations in Escambia County has been mapped.[18]

Wherever the piezometric surface is higher than the land surface, wells that penetrate the artesian limestones will flow. The areas of artesian flow in Florida are shown in Fig. 4. They include three principal areas: the Atlantic Coast, southern Florida, and the Gulf Coast. The map represents the areas of flow only approximately. Within the areas of flow are some relatively high regions in which wells will not flow. On the other hand, flows can be obtained in some low-lying areas that are not indicated as areas of artesian flow.

The height to which the artesian water will rise above the land surface in wells ranges from a fraction of a foot to as much as 65 ft. In the nonflowing wells, in higher areas, the water may stand as much as 100 ft below the ground surface.

Chemical Quality and Temperature

A report by Collins and Howard[19] and a report by Collins, Lamar, and Lohr[20] contain results of chem-

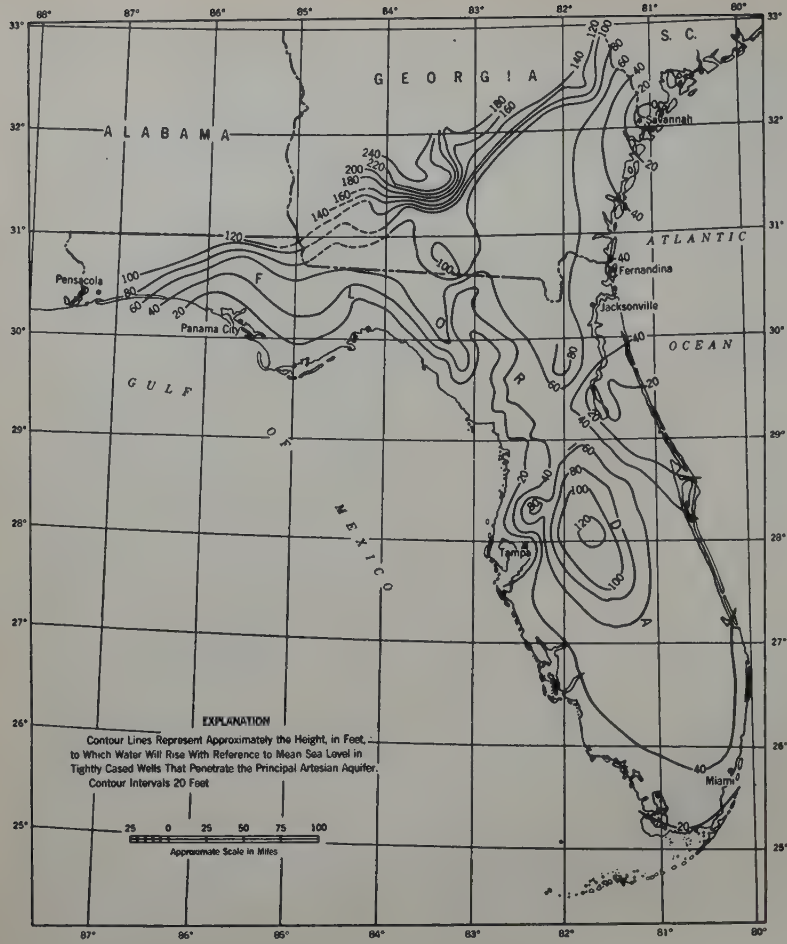

Fig. 2—The piezometric contours in Florida and Georgia.

ical analyses of samples of water collected throughout the State. Other reports[21–23] contain chemical analyses of water in different areas.

In general, most of the water in the Floridan aquifer is typical calcium bicarbonate water such as one would expect to find in limestone. In some parts of the State, as in southern Florida where the limestone contains gypsum, the water is relatively high in sulphate. The hardness of the water from limestone ranges from about 90 ppm in the recharge areas to more than 1000 ppm in water that has moved many miles through the limestone. The artesian water generally contains hydrogen sulphide as it comes from the wells, and hence it is aerated by most of the cities and towns that use it as a source of supply.

Ground water that occurs in sand and gravel is generally much softer than the water in the limestone formations. For example, water from a sand about 200 ft deep at Pensacola has only 41 ppm of dissolved solids and only 12 ppm of hardness. The water in the Floridan aquifer is relatively free from iron, but some of the shallow ground water has as much as several parts per million.

In recent years, as a result of a general recognition of the relation of the fluoride in drinking water to the development of teeth, determinations of the fluoride content of ground water have been made throughout the country. It is now understood that a fluoride content of about 1 ppm is the optimum for dental health; where the content is much less, the occurrence of dental caries (tooth decay) is relatively high, and where the content is greater than 1.5 ppm, the disfigurement known as mottled enamel is prevalent. Only one formation in Florida, the Hawthorn formation, yields water having a fluoride content of more than 1 ppm.[24] The maximum content in this formation, however, is less than 3.5 ppm.[25] The water that has a high fluoride content appears to come from beds that contain phosphatic material, which may be a source of the fluoride. The water from the Hawthorn formation is relatively hard. However, elsewhere in the Gulf and Atlantic Coastal Plains the water that has a relatively high fluoride content is soft and is generally associated with glauconite, bentonite, or volcanic ash. The occurrence of fluoride water in the coastal plain of Alabama is described in a paper by LaMoreaux.[26]

The chloride content of water, which may be used as an indication of the salinity, is especially significant in the coastal areas of Florida, where freshwater supplies may become contaminated by encroachment of salt water, and in a few places in the interior, as along the St. Johns River Valley and in the Everglades at the south end of Lake Okeechobee. Except in the areas where salt-water encroachment has occurred in recent time, or where the formation has not been completely flushed of the salt water that was present when the sediments were deposited or that entered the sediments when the sea stood at a higher level in the geologic past, the chloride content of the water is generally less than 20 ppm, and in many areas it is less than 10 ppm.

Fig. 5 shows the areas in which the artesian water has a chloride content of 100 ppm or more. The available information indicates that the relatively high chloride content, especially in the interior, as in southern Florida and in the St. Johns River Val-

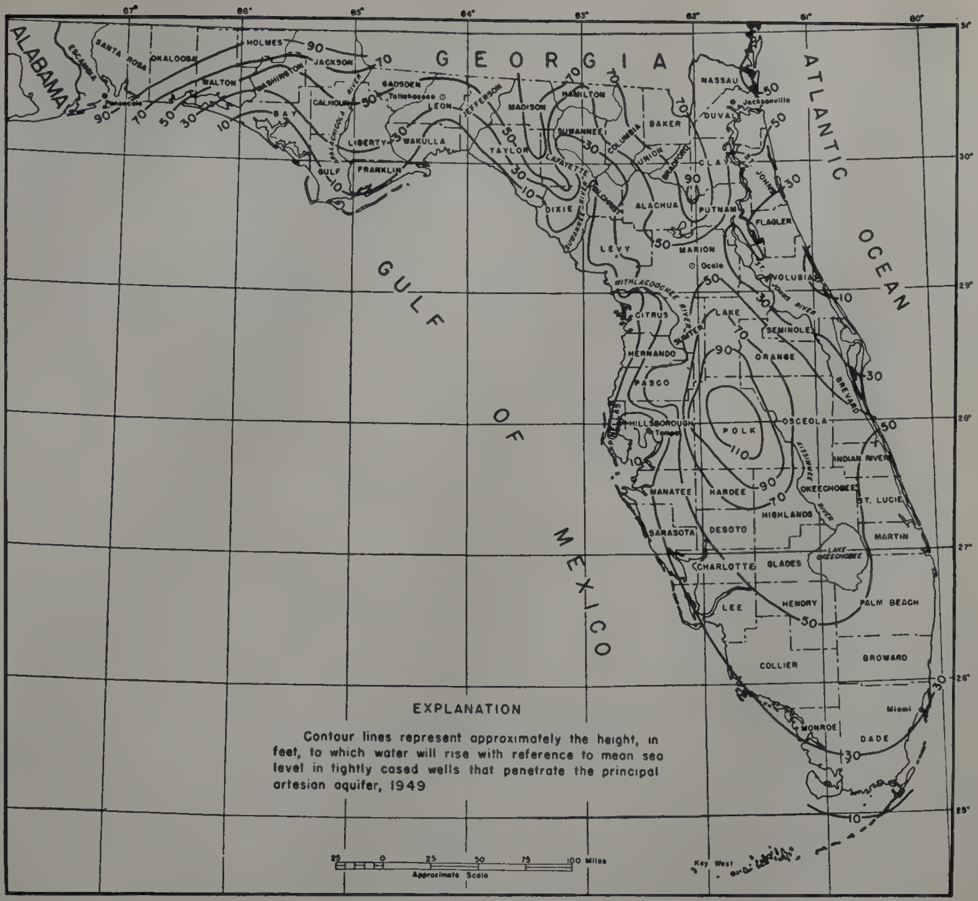

Fig. 3—The piezometric surface in the Florida peninsula.

EXPLANATION

Contour lines represent approximately the height, in feet, to which water will rise with reference to mean sea level in tightly cased wells that penetrate the principal artesian aquifer, 1949

ley, is caused either by traces of salt or salty water that was present at the time the formation was deposited, or by sea water that entered the formation during the Pleistocene epoch when the sea stood as much as 270 ft above its present level.

The temperature of the shallow ground water, as indicated by Collins,[27] is, in general, about the same as the mean annual temperature of the air. Near the ground surface the temperature of the water changes with the air temperature. The temperature of the water increases with depth, as the earth's temperature increases. In general, the temperature increases 1°F for each 50 to 100 ft of depth.

Collins indicates, on a map showing the approximate temperatures of water from nonthermal wells at depths of 30 to 60 ft, that the temperature ranges in Florida from about 70°F in the northern part of the State to more than 77°F in the southern part. Measurements of the temperature of the artesian water at the mouths of the wells in different parts of the State show a range from about 70°F to about 85°F. However, in one well in the southern part of the State, although only 620 ft deep, the water had a temperature of 88°F, which indicates a steeper thermal gradient (an increase of about 1°F for each 33 ft of depth) than that indicated at other wells.

Consumption

The large use of ground water throughout Florida emphasizes the fact that ground water is one of the most valuable minerals of the State. According to data compiled by the Florida State Board of Health,[28] there are about 357 public water supplies in Florida that serve 100 or more people, and of these, about 325 obtain their water entirely from wells. The public supplies serve a total population of 1,980,000, of which 79 pct is served by supplies from wells, and an additional 6 pct is served by supplies obtained partly from surface sources and partly from wells, or from springs and ponds that yield ground water. Practically all the nonpublic domestic supplies in Florida are from wells.

All the larger cities except Tampa, West Palm Beach, and Orlando obtain their water supplies entirely from wells, and Orlando obtains its water partly from wells. Among the larger cities using ground water exclusively are Jacksonville, St. Augustine, Miami, St. Petersburg, Lakeland, Daytona Beach, Gainesville, Tallahassee, and Pensacola. In the early development of the water supply of Tampa, wells were used, but they were near Tampa Bay and were ruined by salt-water encroachment—and so were the first wells of the municipal supply of St. Petersburg. However, by developing a well field at a safe distance from the coast, St. Petersburg has obtained its present excellent ground-water supply. The geologic and hydrologic conditions indicate that an adequate supply of ground water for Tampa could be obtained at a safe distance from Tampa Bay.

The water supply of Jacksonville is unique in that it is the largest municipal supply in the nation that is derived from the natural flow of artesian wells.

In comparison with other States, Florida ranks high in the number of flowing wells used for irrigation and in the value of crops irrigated by flowing wells. According to the U. S. Bureau of the Census,[29] 4631 flowing wells and 1686 pumped wells were being used for irrigation in 1940.

Florida ranks high also in the quantity of ground water used by industries. Among these, the phosphate industry probably uses the largest quantity of ground water. It is estimated that the phosphate companies now operating in the State use an average of about 75 million gal a day. The paper and pulp industry is another consumer of large quantities of ground water. A total of about 65 million gal per day is pumped by the seven paper and pulp mills in Florida, one of which pumps about 30 million gal per day. Other industries that consume large quantities include citrus canning, citrus concentrates manufacturing, ice and cold-storage plants, air conditioning, resinous-products manufacturing, and mining of heavy minerals and limestone.

The total annual consumption, in gallons, of ground water in Florida is estimated to be as follows:

Public supplies serving 100 or more people	57,000,000,000[30]
Industrial supplies	70,000,000,000
Agricultural supplies	35,000,000,000[29]
Domestic supplies	15,000,000,000
Total	177,000,000,000

The total consumption is equivalent to an average of 485,000,000 gal a day. Considering that the value of the water used to be $0.05 per 1000 gal, the annual yield of the ground-water resources of Florida is now valued at about $9,000,000.

The total consumption of ground water is, of course, a heavy draft on the ground-water resources, but it should not be a cause for concern in regard to the State as a whole when it is realized that the ground-water reservoirs are naturally discharging many hundreds of millions of gallons of water a day, much of which can be salvaged and used whenever it is needed. Examples of the discharge are Florida's limestone springs, which rank with the largest in the world. These springs demonstrate the large capacity of the ground-water reservoirs. Among the largest of the springs is the Silver Springs group, which has a discharge that has ranged from 419 million to 756 million gal per day. The average flow of Silver Springs alone, about 500 million gal per day, is equal to the present total consumption of ground water in the State.

Difficulties Involved

However optimistic may be the outlook on the economic aspects of the ground-water resources of Florida, it should be remembered that many of the details of the ground-water geology and hydrology must be determined before an accurate estimate of the total ground-water capacity of the State can be made. The need for this information will increase as the consumption of water and the development of the State increase. Locally, the rate of withdrawal of water may exceed the rate of recharge, and so long as it does, water levels in wells will decline until they ultimately drop below the economic pumping lift or, in areas subject to salt-water encroachment, until the supply becomes contaminated with salt water. In addition to these problems of 1—decline of head and the accompanying decline in yield of wells and 2—salt-water encroachment, there are other complex problems, such as 3—the relation of deeper formations to the overlying aquifers, 4—the amount of leakage from one aquifer to another, and 5—artificial recharge and drainage of aquifers.

Decline of head: Large declines of water levels have occurred in only a few areas, including Fernandina, Jacksonville, and Panama City, where large quantities of water are consumed. In most, if not all, of these areas the rate of withdrawal apparently has not yet exceeded the rate of recharge. However, in each such area sufficient information on the ground-water geology and hydrology should be obtained to show what declines of water levels can be expected with any given rate of withdrawal and to show whether the area is in danger of salt-water encroachment.

A decline in the yield of wells generally accompanies a decline of water levels or artesian head. At Jacksonville and Fernandina, however, the yields of some of the large wells have declined considerably while the artesian head in the surrounding areas has remained relatively stable. This condition indicates that the water-bearing formations around the affected wells have become less permeable. The decrease in permeability is probably caused by precipitation of calcium carbonate in the pores of the limestone as a result of the reduction of hydrostatic pressure around the discharging wells. As yet, however, the phenomenon has not been investigated adequately and hence is not thoroughly understood.

Salt-water encroachment: Salt-water encroachment may occur where the aquifer is exposed to salt water and the water level declines to the extent that salt water may move into the aquifer. The general relations between salt and fresh water in coastal areas have been summarized by Brown.[31] The general conditions in Florida have been described in several reports.[32–36]

The principle of equilibrium between salt water and fresh water, as applied to the hydrology of sea coasts, is commonly referred to as the Ghyben-Herzberg principle,[35] and may be expressed by the formula, $h = t/g\text{-}1$, in which h is the depth of fresh water below sea level; t is height of fresh-water or hydrostatic head above sea level; and g is the specific gravity of the salt water. The specific gravity of sea water varies somewhat from one locality to another and may also vary with depth. If the specific gravity of sea water is 1.025, the fresh ground water will extend 40 ft below sea level for every foot that the ground water stands above sea level. This principle holds only where the salt water is in contact with the aquifer and can move into it when the fresh-water head is reduced.

Salt-water encroachment occurred in the old well fields of Tampa, St. Petersburg, and Miami. At Miami the encroachment was caused chiefly by the lowering of the water table with drainage canals. Salt-water encroachment can be avoided, of course, if the danger is recognized in time and the fresh-water head is maintained at sufficient level to prevent the salt water from entering the fresh-water aquifers.

The Ghyben-Herzberg principle does not apply to some problems of salt-water contamination. For example, in some places the water-bearing formations in Florida contain salt water that is the remnant of sea water present at the time the formation was deposited or of sea water that entered the formation when the sea stood at a higher level than it does at present. Fresh water, moving through an aquifer under those conditions, will become contaminated regardless of what the fresh-water head may be. Such conditions exist in the artesian aquifer in part of the St. Johns River Valley and in the interior south of Lake Okeechobee. Fig. 5 shows the areas in which the chloride content of the water is 100 ppm or more.

Relation between formations: The relation of the deeper formations to the overlying aquifers will be better understood after more wells have been drilled. Only a few of the water wells in Florida are more than about 1000 ft deep. In some parts of the State

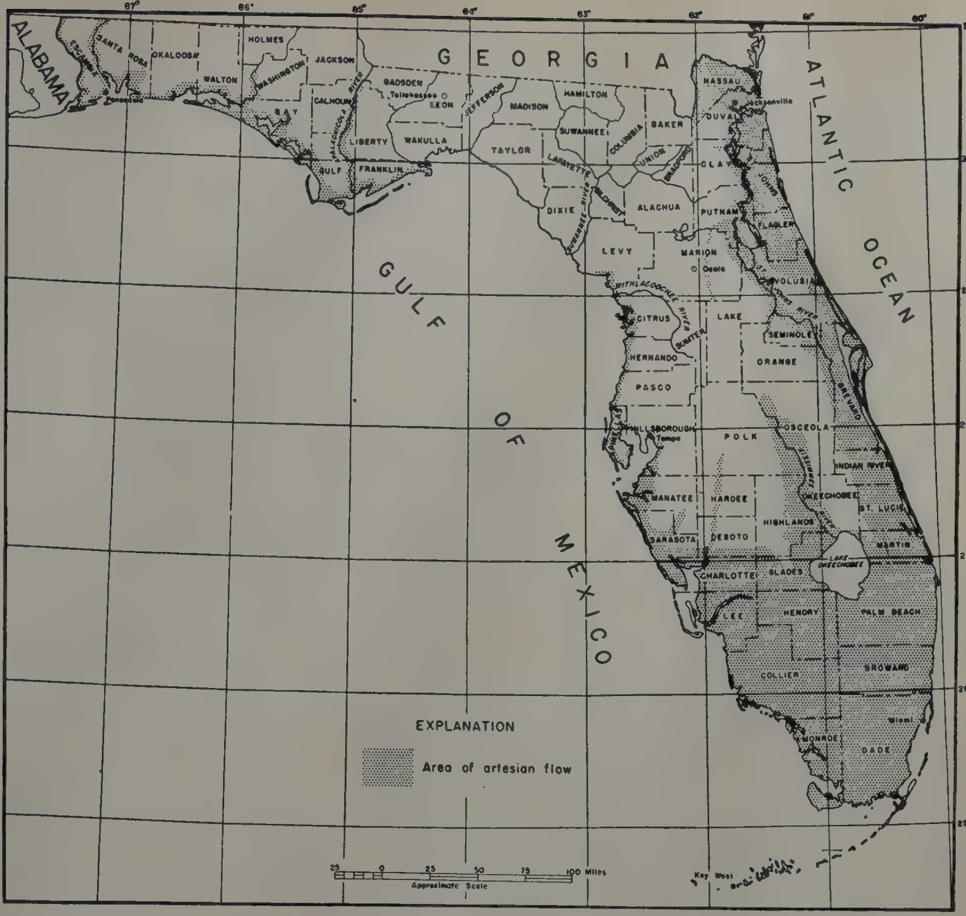

Fig. 4—The areas of artesian flow in Florida.

additional ground-water supplies have been obtained from depths of more than 1000 feet. However, drilling to great depths should be done with care, because in much of the State the deeper formations contain salt water, some of which is under higher head than the fresh water. Careless drilling into salty water might, therefore, contaminate the fresh water supply.

Leakage from one aquifer to another occurs at some places where the water in one water-bearing bed is under higher pressure than that in the overlying or underlying beds. Under these conditions water may leak directly from one bed to the other or may move through uncased wells or through wells having defective casings. Only a few of the artesian wells in the peninsula are completely cased, and in many of the wells the open holes extend through several water-bearing beds. Thus, part of the water entering the wells from the beds that have relatively high pressure may escape into beds having lower pressure. Observed differences in head between water in the Ocala limestone and that in overlying beds of the Hawthorn formation range from a fraction of a foot to as much as 40 ft. Such leakage is known to occur in parts of the State, and it will occur over more of the State if new wells are not properly cased. Although no underground leakage was revealed in the exploration of wells with a deep-well current meter in Sarasota County,[36] it is entirely possible that some leakage had taken place but that the pressure of the water in the two horizons had reached equilibrium by the time the tests were made.

Artificial recharge and drainage: Wells are used at some places to drain surplus surface water into water-bearing formations, and at other places, where mining or construction operations require it, to unwater or drain shallow formations.

Surplus surface water is drained into wells in some parts of the lake region and in other areas where surface drainage is poorly developed and inadequate. At these places the water levels in wells stand several feet below the surface and surface water is drained into wells constructed for that purpose. The largest number of drainage wells are in Orange County, where the wells penetrate the Eocene limestone; and in Dade County, where the wells penetrate Pliocene limestone. In addition to the sanitary problem relating to these wells, there is also the problem of the capacities of the wells and of the aquifer to receive water, as discussed in several reports.[37-41]

If drainage wells operate under gravity flow, as they ordinarily do, the static water level in the wells must, of course, be below the level of the water being drained. Other factors being equal, the capacity of a well to receive water is more or less in direct proportion to the distance betwen the two water levels. Some wells drilled for drainage have overflowed under artesian pressure during wet seasons, thus aggravating the floods they were calculated to prevent. Therefore it is evident in this problem, as in the others, that adequate information on the geology and hydrology should be available to show the character of the water-bearing material, the occurrence of the water, and height to which the water will rise in the well, not only at the time the well is drilled but throughout the year, because seasonal water-level fluctuations of more than 10 ft may occur.

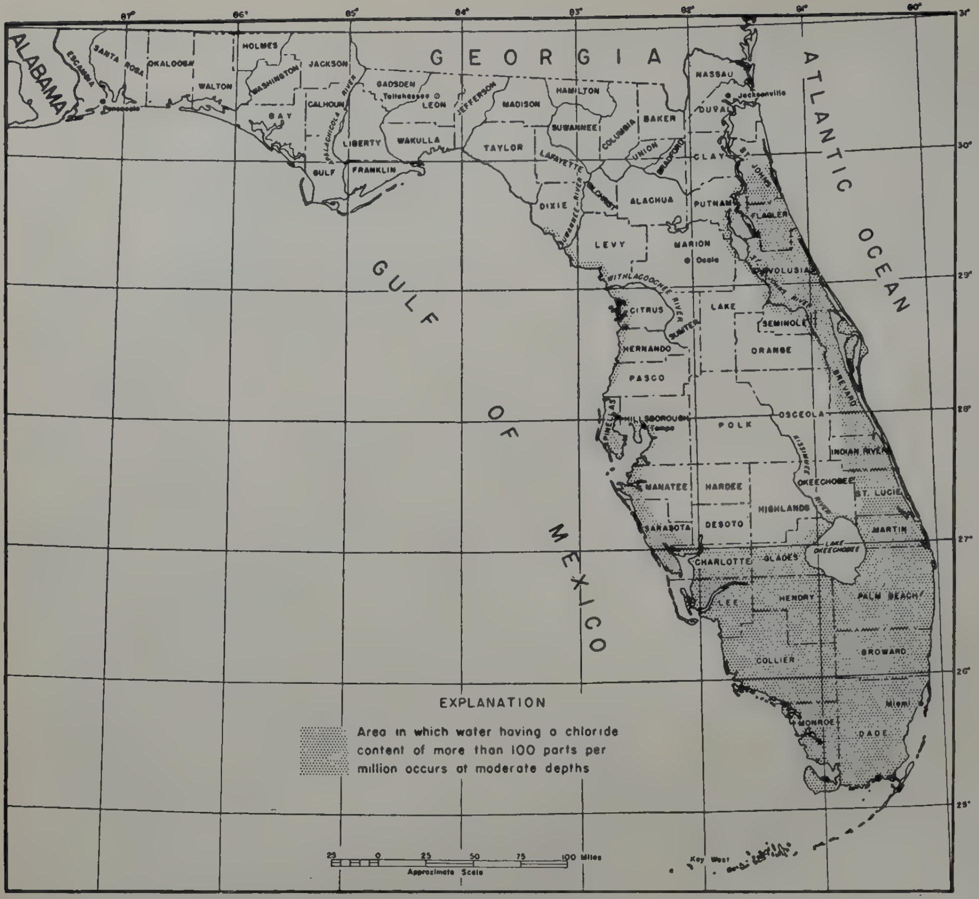

Fig. 5—Areas in which chloride content of the water is 100 ppm or more.

In some mines high ground-water levels are a serious problem, and a study of the ground-water conditions is needed to determine the most economical means of controlling the water. Some of the problems in the mines in the Southeastern States are discussed by DeBuchananne.[42]

Other drainage problems require special ground-water studies in order to determine the correct solution. A unique problem of this kind arose during World War II in connection with a project at Hookers Point, near Tampa. The project called for the pouring of concrete hulls for ships in basins excavated below sea level, on a point of land adjacent to a body of sea water. The basins in which the hulls were to be poured were dug in land to be drained by pumping through sand points, and a reliable estimate of the required rate of pumping was needed in order to determine whether the project was feasible. On the basis of pumping tests to determine the permeability of the formations, the junior author gave an estimate which later was found to agree closely with the actual pumping rate.

In some areas surface drainage through wells or pits or by water spreading is beneficial in recharging aquifers in which the rate of withdrawal may exceed the rate of natural recharge. Artificial recharge of ground-water reservoirs—largely by water spreading—has been practiced at many places in the Western States[43] for many years. In the East, artificial recharge has been practiced at Runyon, N. J.,[44] for more than 40 years.

On Long Island, New York,[45] in 1944, more than 200 recharge wells and several recharge basins were in use for recharging the underlying aquifers at a summertime rate of about 60 million gal per day.

In central Florida,[46] the drainage of surface water into wells recharges the aquifers, although the purpose of the wells is to dispose of the surface water. The aquifer in Orlando and vicinity has the capacity to receive large quantities of water through wells, because the aquifer is very permeable and contains many solution channels. As artesian aquifers have very small capacities for storing additional water, the effects of artificial recharge in them disappear rapidly during periods in which the rate of recharge is light. Under these conditions a continuous supply of water for recharge during drought must be available if the piezometric surface is to be maintained at a high level.

Conclusions

The large consumption of water for industrial, municipal, domestic, and irrigation use emphasizes the fact that ground water is one of the most valuable of the industrial minerals of the State. It is estimated that 177 billion gal of ground water is used annually.

Along with an optimistic outlook on the economic aspects of the ground-water supply in Florida, it should be remembered that many of the details of the ground-water geology and hydrology are yet to be determined in order to estimate accurately the total ground-water capacity of the State. Some of these details involve complex problems, such as that of salt-water encroachment in some of the coastal areas.

It is evident that, although there is a general abundance of ground water in Florida, there is nevertheless a need for adequate investigations to provide information required for wise and careful planning of its development and conservation. The solution of the problem requires the combined efforts of engineers, geologists, chemists, and other investigators, and the cooperation of the well drillers and owners of wells.

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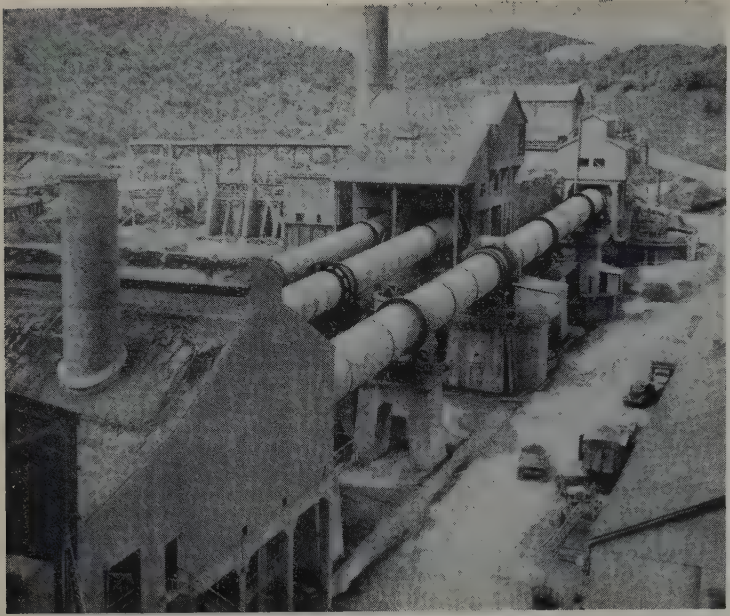


Fig. 1—Calaveras Cement Co.

The Calaveras Cement Co. Dust Suit

by Wm. Wallace Mein, Jr.

IN March 1949 the Calaveras Cement Co. was sued by five landowners whose properties are located in the vicinity of the plant. These landowners—all of them cattle ranchers—sued for dust damages of $120,338 and for an injunction preventing the company from casting dust upon their properties in injurious quantities. The issue was split, the jury deciding the amount of damages and the court handling the injunction features. The action came to trial before a jury in April of that year and resulted in damages being awarded in the amount of $7,508. A month later the court issued its injunction requiring recovery of stack dust to an 87 pct minimum.

The Calaveras plant, shown in Fig. 1, is located three miles from San Andreas in the heart of the Mother Lode country, about 45 miles east of Stockton, Calif. Since it is in a small town, the company's responsibilities to its employees and to the community are different from those of plants located in large centers of population. Construction of the plant was begun in 1925, and production was started the following year. Standard gray Portland cement as well as several specialty cements are produced.

Production facilities have been continually enlarged so that the present capacity is more than double what it was 25 years ago. The most important item in the company's expansion program was the purchase in 1945 of an 11 ft 3 in. x 360 ft Allis-Chalmers kiln which had been declared surplus by the Defense Plant Corp. The company is now able to produce about 2,500,000 bbl of cement a year, and employment at the plant is approximately 300.

The production of cement has always involved a dust problem, solution of which has been attempted in a number of different ways. By far the most efficient method of catching dust discharge is electric precipitation.

Soon after the company went into production in the 1920's, the landowners near the plant began complaining about dust discharge. The company settled the matter by paying them a total of $27,000 for dust damages incurred during the period from 1929 to 1938. In the latter year the two-unit Cottrell Electric Precipitator shown in Fig. 2 was installed and collected over 85 pct of the dust resulting from operation of the two 11 ft 3 in. x 10 ft x 240 in. kilns. From 1938 until 1946 the landowners registered no objection and were apparently satisfied with that dust recovery. In 1946, however, the substantially higher production resulting from operation of the new kiln immediately overtaxed the facilities of the existing two-unit precipitator.

In a short time claims for dust damages were again received from the landowners. The company informed them that because of postwar shortages it would take several years to install a new precipitator. The company stated that it intended to purchase an additional precipitator and offered in the meanwhile to pay the landowners a fair amount to cover their damages. This offer was rejected and the landowners filed suit in the Superior Court of Calaveras County asking for total damages of $120,338 and for the injunction outlined above.

Case for the Ranchers

The attorneys for the plaintiffs produced three different types of witnesses: first, the landowners themselves and members of their families; second, other cattle raisers who were sympathetic to the plaintiffs; and third, several types of technical experts: veterinarians, soil chemists, and an aerial photographer. The testimony of these witnesses was designed to establish that the flue dust from the plant had not only damaged the land and the forage, but had also caused fluorine poisoning to their animals. By a combination of these causes, the plaintiffs attempted to prove substantial loss of profits.

WM. WALLACE MEIN, JR., Member AIME, is Vice President, Calaveras Cement Co., San Francisco.

Discussion on this paper, TP 3047H, may be sent to AIME by July 31, 1951. Manuscript, Nov. 18, 1949; revision, Feb. 2, 1951. Los Angeles Meeting, October 1949; New York Meeting, February 1950.

The plaintiffs introduced into evidence a mosaic map made from a large number of photographs taken by their aerial photographer at an altitude of 7500 ft above sea level, or approximately 6500 ft above the terrain in the area of the plant. The plaintiffs themselves, and virtually all the other witnesses too, told of heavy dust deposits which reduced the carrying capacity of the acreage and made their cattle sick, emaciated, and nonproductive.

Dust gaging stations were installed on the properties of all the plaintiffs and dust samples collected on vaseline-coated glass plates. The soil chemist hired by the plaintiffs also took soil samples, analysis of which, he testified, showed pH values above 8.0 in a majority of the cases, with a high in one sample of 9.7. A pH of 8.0 to 8.5 is excessively alkaline to the point of causing sterility in the ground. As a result, according to the plaintiffs' witnesses, there was a noticeable lack of the low growing grasses, which are the best feed.

The attorneys for the plaintiffs placed heavy reliance on the testimony of three veterinarians who stated that they had examined the animals of all five of the plaintiffs and had found definite symptoms of fluorine poisoning. Fluorine poisoning, they testified, was accumulative and progressive: Once it had advanced to the stage found in the plaintiffs' animals there was no hope of future recovery.

Claiming that the carrying capacity of the land had been greatly reduced by the dust, the plaintiffs said that they were forced to curtail drastically the size of their herds. Because of fluorine poisoning, their calve crops were greatly reduced. And finally, many of the remaining cattle were so stricken with fluorosis that they were not marketable as beef and could be sold only to the tallow factories. This situation was pictured in the framework of a cattle market in which the price of calves had doubled in value between 1945 and 1948. It was their contention that damages should include a figure representing a capitalization of their lost profits.

They maintained in addition that they were entitled to further damages for nuisance and inconvenience suffered in their residences. The ranchers' wives who lived on their properties stated that the dust doubled the amount of housework, covered their food, clothing and furnishings, and made living conditions intolerable.

Case for the Cement Company

The company's attorneys had anticipated and made preparations to answer most of the contentions in the plaintiffs' case. Testimony was prepared to contradict their claims for damages to the land, forage and cattle. The charge of fluorine poisoning, however, was a complete surprise. It was necessary to prepare in a few days, testimony which would refute this charge by the landowners. In defense of this charge Calaveras used as its consultant Professor Emeritus Robert E. Swain of Stanford and as its expert witness, a toxicologist, Dr. Charles Hine.

The company's other witnesses were similar to those produced by the plaintiffs—veterinarians, chemists, and soil analysts. The company also put cattle ranchers on the stand, and in addition, a local real estate broker, the plant superintendent, the assistant superintendent, and one of the plant engineers.

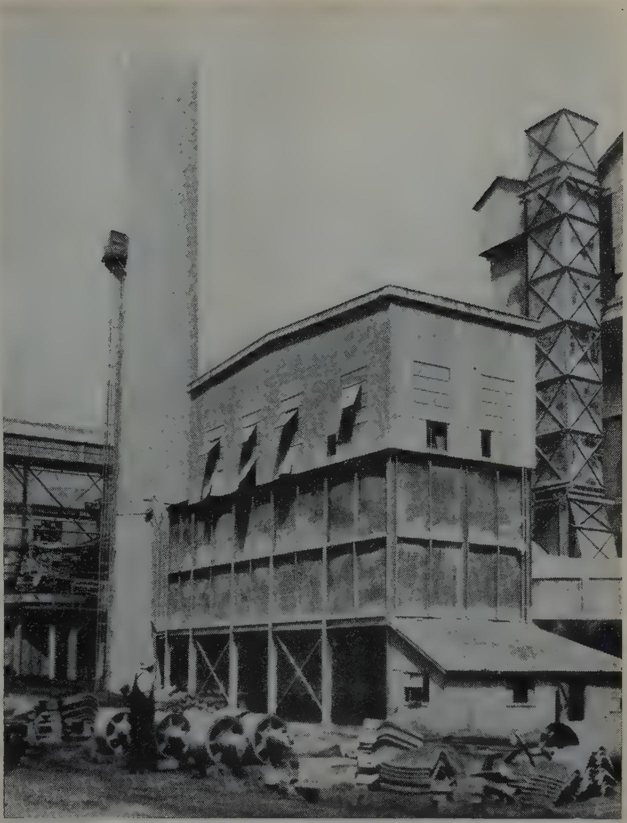


Fig. 2—Two-unit precipitator installed in 1938.

The company engineer had spent several months before the trial investigating the dust areas in the vicinity of the plant. An aerial photographer was employed to take pictures of this area at an altitude of 2500 ft above sea level, or 1500 ft above the land's surface. From these photographs and from observations made on foot by the engineer, a dust survey of the area was prepared on a blown-up U. S. Geological Survey map. The acreages of each of the ranchers were segregated into five composite dust zones—(1) very heavy, (2) heavy, (3) medium, (4) light, and (5) very light.

Calaveras introduced into evidence an exchange of correspondence between its attorneys and the plaintiffs' attorneys in which a request to erect test stations on the ranchers' properties was refused. The company then produced a legal order granted by the court giving its witnesses the right to visit and examine the properties.

Several months before the trial Calaveras had hired an agronomist to make a survey of the plaintiffs' lands and their cattle. Photographs and slides which he exhibited showed superior forage crops and fine specimens of cattle.

The agronomist took forage samples and soil samples from six different locations on the plaintiffs' lands. Analysis of the soil specimens indicated that the pH values on the ranchers' lands were just about neutral. The extreme low was 7.13 and the extreme high 7.85. He stated that operation of the cement plant had not damaged the soil or the surface of the

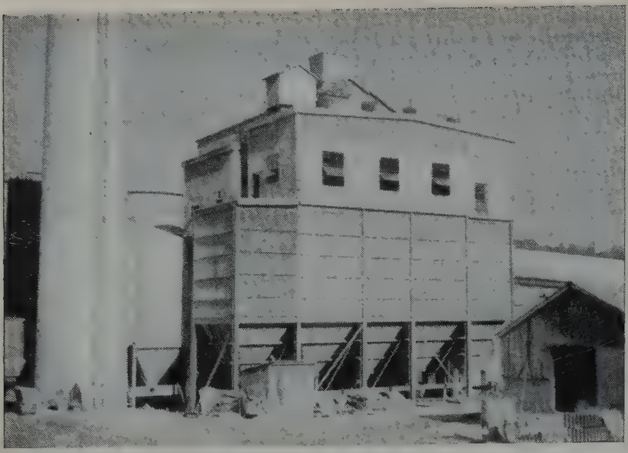


Fig. 3—Precipitator installed in 1949.

plaintiffs' lands either on the basis of plant growth or of chemical analysis. He admitted that the forage crops had become covered with dust which made them less palatable during the dry season, but he stated that this dust actually benefited growth of clover and other choice grasses.

The testimony of the company's next two witnesses, its Chief Chemist and the Chief Chemist for Twining Laboratories at Fresno, refuted the principal contention of the landowners. The Calaveras witnesses went to the three most important gaging stations used by the plaintiffs for collecting dust on the vaseline-coated glass plates. Here, on the ground around the same telephone poles put up by the plaintiffs for their own dust determinations, the company chemist gathered samples of both forage and soil and sent them to the Twining Laboratories for analysis. Then, for complete contrast, samples were taken at a fourth point 15 miles away from the dust zone. The Twining chemist found that the grass samples taken at this location actually contained 20.9 parts per million of fluorine, compared to 3.8, 9.5, and 13.3 parts per million in the samples taken at the landowners' gaging stations. Analysis of the soil samples revealed that the soil at the 15-mile point contained 59.5 parts per million of fluorine as against an average of 7.8 parts per million in the samples found underneath the three gaging stations.

Dr Hine stated that he had examined 16 of the plaintiffs' animals several days before and had found absolutely no sign of fluorosis. His calculations showed that an animal would have to consume 15 times its normal intake of 30 lb of food per day to obtain a toxic amount of fluoride, namely, 200 parts per million.

In order to establish a reasonable figure for loss of rental value, the company produced a local real estate broker and several ranchers in the community, all of whom testified that they were familiar with the land values of the five ranchers compared to values of other property in the area. These witnesses stated that the plaintiffs' lands and forage crops compared quite favorably to similar grazing property outside of the dust zone. They confirmed the agronomist's statement that the best food grew on the land with the heaviest dust deposits. Each of these witnesses calculated what he thought was a reasonable loss of rental value for the five rancher plaintiffs. Their estimates averaged $4,600 total loss of rental value during the 4-year period of dust damages.

In summing up the case, Calaveras's attorney was frank to admit that the plant had created a dust nuisance during the period in question. He stated, however, that the extent of this damage had been enormously exaggerated by the plaintiffs and should be limited to the much lower loss of rental value as calculated by the witnesses. He emphasized the spurious nature of the fluorine claim made by the plaintiffs. Finally, he stated that the plaintiffs' animals which the company's witnesses were allowed to examine were in poor condition because of old age, lice, and an adverse winter, rather than because of any fluorine poisoning from their forage.

Verdict and Aftermath

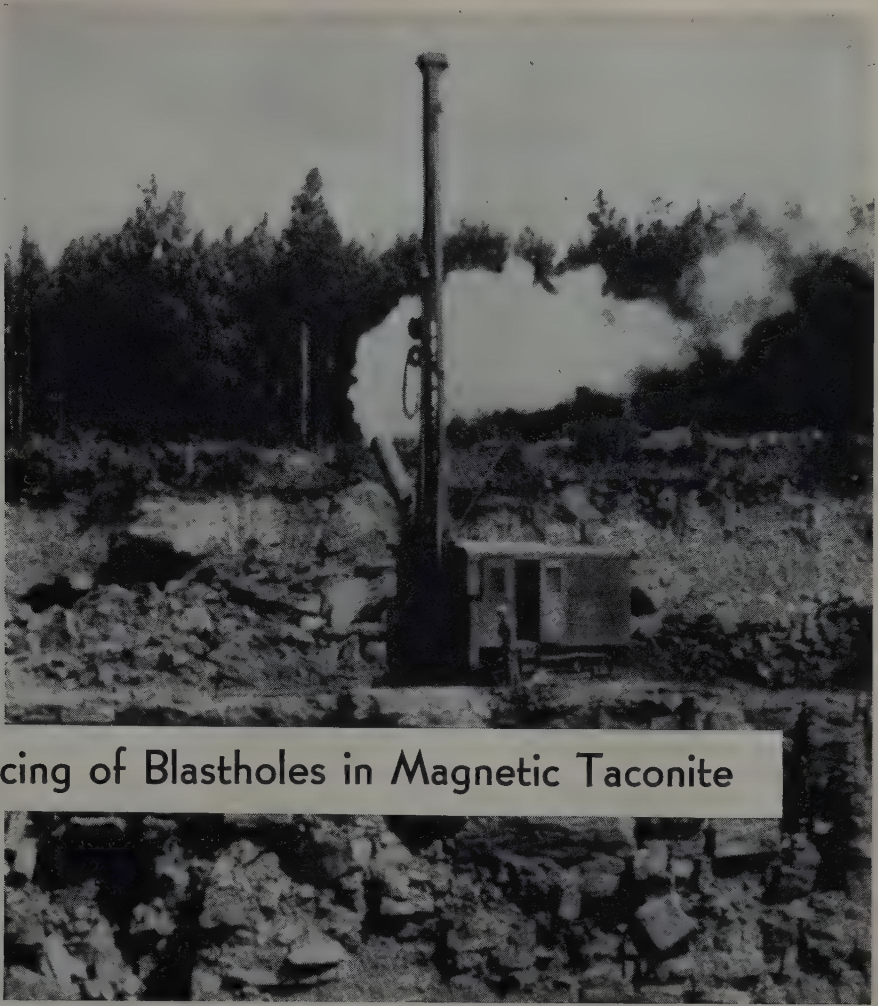
Two weeks after the opening of the trial the jury brought in its verdict awarding the ranchers a total of $7508 for damages, instead of the $120,338 for which they had sued.

The award of damages, however, was only half of the suit. The other portion of the suit was the request for an injunction which the judge announced at the outset would be decided by him later without benefit of the jury. A month after the jury verdict, the judge held a hearing on the injunction proceedings. The Calaveras plant superintendent pointed out the importance to the community of the continuing operation of the cement plant. He testified that the 1948 annual payroll to Calaveras County residents was close to a million dollars and that local trucking operators secured an additional half a million dollars worth of business from the company each year. Finally he showed that to enjoin the company from discharging any dust at all would shut down the plant, thus resulting in a far greater loss to the stockholders than the limited loss to the landowners in the partial dust discharge.

After hearing the testimony, the judge requested that each side prepare what it considered a proper order. The proposed draft which Calaveras presented stipulated recovery of not less than 87 pct of all flue dust that would be discharged from the kilns in the absence of any control. In determining the amount of dust discharge, the company recommended use of the method described in Bulletin WP50, entitled "Method for Determination of Velocity, Volume, Dust and Mist Content of Gases," published by the Western Precipitation Corp. The proposed injunction prepared by the plaintiffs' attorneys provided that the cement plant would be permanently enjoined from operating in such a manner as to cause injury to the soil, vegetation, cattle, or homes of the five plaintiffs.

Shortly, thereafter, the judge adopted an order and injunction along the lines of the draft submitted by the company. A few days prior to June 30, the grace period, the company's new precipitator was placed in operation. The Western Precipitation Corp. had assured that the new precipitator, Fig. 3, operating in conjunction with the existing one would have no difficulty in collecting 87 pct of the flue dust discharge. The total cost of installation was approximately $185,000. The annual operating cost for the precipitators is $26,000 before depreciation and $47,000 including depreciation.

Later that year, the plaintiffs' motion for a new trial was denied by the court. However, in the following month, the plaintiffs filed an appeal in the case. It was their contention that their own version of the injunction asking that the plant be prevented from discharging dust in injurious quantities should be the one adopted by the court. This appeal is still pending.



Production Jet-Piercing of Blastholes in Magnetic Taconite

by D. H. Fleming
and J. J. Calaman

DURING 1950 the jet-piercing process was used commercially in the piercing of primary blastholes in magnetic taconite at the preliminary taconite plant of the Erie Mining Co., Aurora, Minn.

The Erie Mining Co. purchased a jet-piercing machine and associated equipment in 1949, and the machine has been in operation since its installation in the summer of that year. An overall view of the pit with the JPM-1 jet-piercing machine in operation is shown in Fig. 1.

During three months of 1949 and continuously since February 1950, production piercing took place under all pit conditions. By the end of October 1950, a total of 15,000 linear ft of hole had been pierced in an open pit handling an average of 1200 to 1800 tons of crude a day in one 8-hr shift. While it is recognized that this output does not approach the millions of tons a year planned for the developing Minnesota taconite projects, the machine as a unit has been operating under conditions probably more adverse than those that eventually will be encountered in the large operations. This pilot operation has permitted evaluation of the piercing problem in the full-scale taconite plant.

This paper describes jet-piercing process fundamentals, important features of the latest jet-piercing burners and machine, actual piercing results obtained, and new developments that give promise of materially increasing overall process efficiency.

To produce the blasthole, jet-piercing utilizes thermal energy, as contrasted with the application of mechanical energy in churn drills and jackhammers. The heart of the process may be termed a tailored flame produced by a rocket-type burner, thermodynamically similar in every respect to that of the giant rockets being developed for military purposes. The heat release in the combustion chamber of the burner is about $50\text{x}10^6$ Btu per cu ft per hr as compared with $5\text{x}10^4$ for most modern steam generators.* Combustion of liquid petroleum fuels (kerosine, diesel oil, or No. 2 fuel oil) with gaseous oxygen in the burner chamber at pressures of 100 to 200 psi results in temperatures of about 4300°F.

Expansion of the hot combustion gases through divergent nozzles produces high-temperature supersonic jets having measured velocities of about 6000 fps. Impinging the flame jets on rock causes a thin surface layer to expand and break away from the base as a result of the thermally induced stresses. This spalling action is aided further in chert and quartzite by the 0.82 pct volume change resulting from the δ to β quartz inversion at 1066°F.† The dynamic action of the jets whisks the spalled rock

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* John H. Perry: Chemical Engineer's Handbook (Third Edition), p. 1639.

† R. B. Sosman: The Properties of Silica, Chap. XX.

This page of Transactions is a continuation from p. 536. The missing pages appeared in Mining Engineering.

Fig. 1—The pit at Erie Mining Co., Aurora, Minn., with the JPM-1 jet-piercing machine in action.

out of the path of the flame and consequently exposes fresh surface to attack. Some fusion is likely to occur in broken rock subsurface or where the iron band is thick and consequently in greater concentration, but the depth of melt is shallow and is immediately blown away, quenched, and embrittled by the steam and water, broken up if need be by the hole sizer, and easily ejected. Slower moving flames, although capable of spalling rock, lack the energy necessary to remove initial spallings and thereby to expose continuously new rock surface. Consequently, operation of the piercing process depends on the combination of high heat transfer rates for rapid spalling and high kinetic energy of the flame to keep the cutting face clean.

Water, a vital ingredient of the process, serves three purposes. 1—It acts as a coolant to preserve the combustion chamber and nozzles. 2—The cooling water is discharged near the bottom of the hole where it quenches and embrittles any material that may have been fused by the flame. 3—The waste heat in the combustion products flashes a large portion of the water to steam, which in turn functions as the conveying media for lifting or ejecting the cuttings continuously from the hole. The combustion gases alone could not do this job. The appropriate ratio of steam to combustion gases is 8:1.

Piercing Technique

The burner is mounted on the lower end of a blowpipe, the length of which is slightly greater than the maximum depth of hole to be pierced. As it is advanced into the hole the blowpipe is rotated to expose uniformly the rock in its path to the action of the flames. The rotation also helps to maintain a straight hole. If ore or rock to be pierced were uniform in composition and solid in formation, the blowpipe advance could be synchronized directly with the spallability of the rock for a given burner. If such were the case, constant high piercing rates would be obtained. For example, when solid sections of taconite are encountered, the piercing rate has been increased to as high as 60 ft per hr. However, this ideal condition is never obtained in normal pit conditions for three reasons: 1—The ore is not uniform in character, since there are alternate bands rich in silica or in iron; 2—There are holes, natural cracks, and cleavage planes; 3—Blasting will produce backbreak that must be passed through. The rock structure discontinuities lower the piercing rate and require control mechanisms to indicate their presence and special operating techniques to pierce them.

To illustrate the effect of discontinuity in the rock, consider the simplest case, an open horizontal crack or seam as shown in Fig. 2. In solid rock, *A*, the bottom of the hole directly under the flames generally assumes a hemispherical shape when using the conventional-type burner, which has one flame parallel with the blowpipe axis and one or more flames at an angle with the axis. The advancing hole, *B*, breaks into the crack with a small hole. The axial flame will jump the gap and start working on the adjacent rock face, the angle flames will continue to spall the triangular wedge of rock but at a reduced rate. Spalling does not readily take place on sharp corners or acute angular areas; in fact, in some instances, melting will occur. The blowpipe will advance until the end of the hole sizer starts to ride on the restriction, or collar, as it is commonly called. This causes part of the blowpipe weight to be transmitted to the collar; tension in the blowpipe suspension cable is reduced correspondingly over normal cable tension when the blowpipe is hanging free.

An instrument in the operator's cab constantly indicates cable tension. When cable tension drops below a critical point, which occurs when a collar is encountered, the operator stops the blowpipe advance, then retracts the blowpipe, *C*, a specific predetermined distance, 2 to 5 in., depending upon the particular burner design. The blowpipe is held at the retracted level for 10 to 30 sec while the angled flames burn out the restricting collar. The blowpipe then is advanced to the point of maximum advance and, if no drop in cable tension is noted, blowpipe advance, *D*, is continued; otherwise, the collar cutting cycle is repeated. If the operator finds that collars are occurring too frequently, the advance rate of the blowpipe will be cut back. On the other hand, when the blowpipe is running free, the operator will gradually increase the advance rate to achieve a high average piercing rate.

An ammeter in the rotary drive motor circuit also keeps the operator informed of developments in the hole. Nominal increases in rotary drive torque occur when a collar is encountered. Sometimes rubble will cave in on the blowpipe, causing sharp increases in rotary drive torque, although there will be no decrease in cable tension. Under this circumstance, the operator will switch from low, 15 rpm, to high, 45

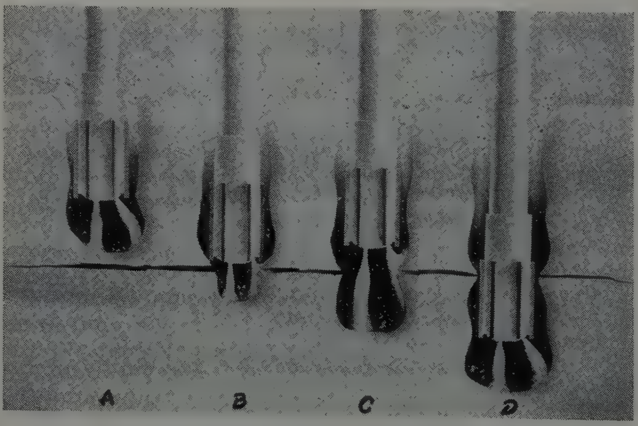

Fig. 2—Piercing through rock with a discontinuity—an open horizontal crack or seam.

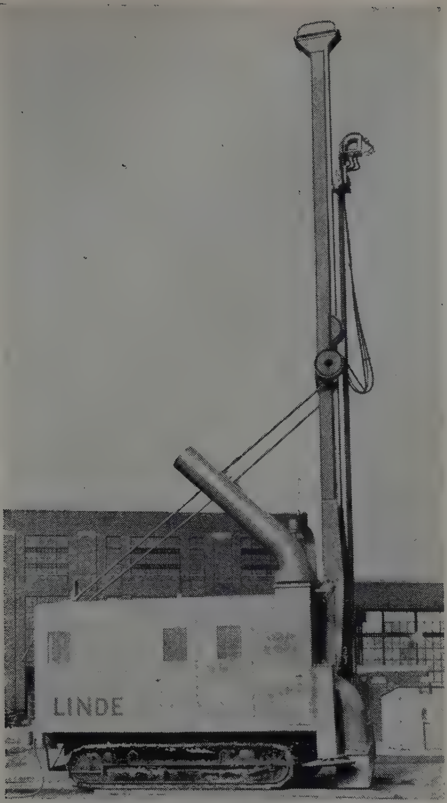

Fig. 3—The first JPM-1 jet-piercing machine.

rpm, to grind up the rubble. These two instruments enable the operator to judge at all times what is occurring in the hole and control the blowpipe advance accordingly.

The primary purpose of a piercing machine is to provide a mechanism for suspending, rotating, and controlling the advance of the blowpipe. In addition, the machine must be capable of rapid moves between holes and quick setup and leveling; it must provide means for metering and controlling the flow of oxygen, fuel, and water and provide means for exhausting the effluent gases from the top of the hole.

The first production jet-piercing machine, shown in Fig. 3, is now being used by the Erie Mining Co. The power supply is 440-v, three-phase, alternating current brought to the machine through a trailing cable according to conventional open-pit practice. The schematic machine cross-section in Fig. 4 shows the major components. The totally enclosed cab is mounted on caterpillars, which give a fixed forward or reverse speed of ¾ mph and are driven by a 40-hp motor. The 48-ft mast, kept in a vertical position for all but the longest moves, is raised and lowered by a winch actuated by the main drive motor. On the forward side of the mast are track guides that stabilize the blowpipe in its vertical travel. Hydraulic jacks ensure rapid leveling of the machine after it has been positioned over the hole marker. The average time for a hole-to-hole move, including leveling, is 15 min.

One of the most important machine components is the blowpipe winch, which pays off the blowpipe at a controlled speed and provides a means for indicating cable tension. A high and low speed range is provided. The low-speed range, which is infinitely variable between ¼ and 24 in. per min, is used for blowpipe advance. The usual operating range is from 3 to 6 in. per min. The high-speed range is used for backing away from collars and retracting the blowpipe from the hole. Although a high-speed range of 20 to 60 fpm is provided, the high speed is generally held fixed at about 30 fpm.

The blowpipe passes through and is rotated by the rotary table by means of drive pins that slide in the blowpipe flutes as the blowpipe moves up and down. Rotational speeds of 15 and 45 rpm are provided. The lower speed is used for normal operation, as previously mentioned, and the high speed is used in a tight hole or when rubble falls behind the blowpipe.

Jet-Piercing Blowpipe

The blowpipe, shown in Fig. 5, consists of three main components: the swing joint, kelly, and burner assembly. Oxygen, water, and fuel are introduced into the rotating blowpipe through the swing joint. This vital part of the assembly is precision-machined of selected grades of stainless steel to give adequate corrosion resistance and minimum wear. The suspending bail is attached to the main thrust-bearing housing, which is located immediately below the swing joint. The blowpipe guide blocks, which ride in the mast track, are attached to the thrust-bearing housing. The round, triple-fluted kelly is standard oil-well drilling equipment.

Oxygen and fuel are carried through separate tubes on the inside bore of the kelly to the burner assembly. Water is brought to the burner in the remaining space inside the kelly. The burner assembly consists of four main parts: the header, fuel injector, burner, and hole sizer. The header is a permanent part bolted to the bottom of the kelly onto which the fuel injector and burner are screwed. The liquid hydrocarbon fuel is atomized by kinetic impact of the gaseous oxygen as it enters the rear end of the pear-shaped combustion chamber. The burned gases are exhausted through divergent nozzles to produce the piercing jets. Oxygen and fuel flows are 10,000 cu ft per hr and 40 gal per hr, respectively.

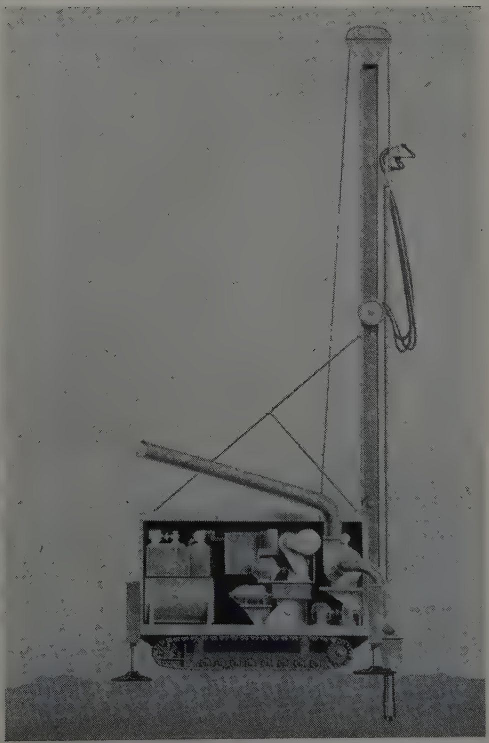

Fig. 4—Cross-section of the jet-piercing machine.

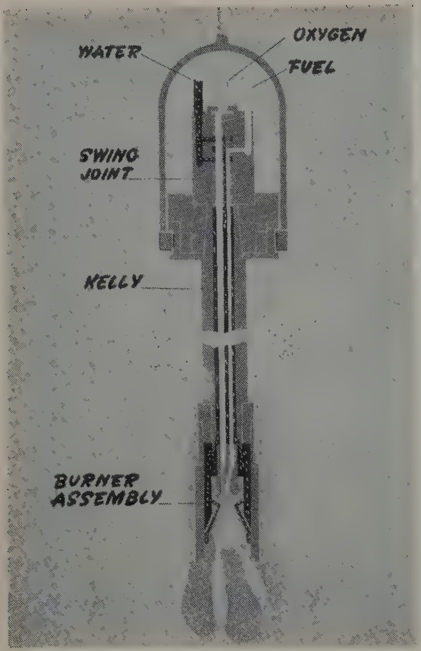

Fig. 5—The blowpipe consists of three main components, the swing joint, kelly, and burner assembly.

These flows have been fixed as the optimum for piercing the 6½-in. minimum diam holes now being made. If the flows are dropped substantially below this figure, the piercing rate is decreased to the point where labor and equipment costs are too large in the overall cost picture. Above these flows, commensurate increases in piercing speed have not been obtained.

The hole sizer, functioning also as a water jacket, encloses the header, fuel injector, and burner. It is constructed of heavy-wall steel tubing. Longitudinally welded to the outside surface of the hole sizer are heavy steel ribs or teeth whose top and front surfaces are coated with Stellite hard-facing material. The ribs extend a greater distance beyond the end of the hole sizer tube than the burner so that the burner front cannot be carelessly wiped on the bottom of the hole. The hole sizer takes the brunt of abrasions in the hole from cuttings, rubble, and scraping on collars. The Stellite alloy on the ribs is touched up after 70 to 90 ft of hole and brought up to gage on the front end where most of the wear takes place. After 11 to 12 hard-facings with Stellite alloy, the front end of the hole sizer shows some heat checking, which can be remedied by a light machining cut or welding in a new front section. The cooling water passes between the inside of the hole sizer and the outside of the burner and is discharged across the front face of the hole sizer. By so locating the cooling water discharge, the steel hole sizer is not damaged by backwash of the flames. The hole sizers are removed from the kelly by a power wrench.

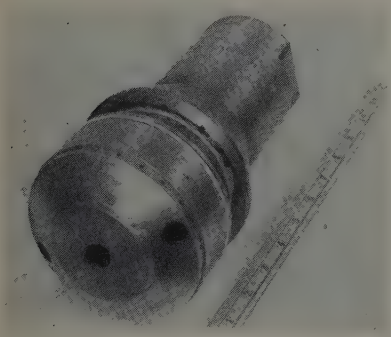

Fig. 6—Triple-orifice burner used in 1950.

The jet-piercing burner is the key to good process economics. The number, location, and angularity of the flames with respect to the blowpipe axis, together with lesser variables such as combustion chamber shape and fuel injector design, have been the subject of much development work to achieve the most efficient combination of variables. So far, a total of 25 different burner types have been designed and tested. The most satisfactory has been the triple-orifice variety shown in Fig. 6. This type of burner has been in use for most of the 1950 season. It has shown an overall cost reduction of 20 pct in terms of cents per ton of ore in front of the shovel over 1949 designs because it will pierce a larger hole, 6½ in. against 6 in., at no sacrifice in speed and no increase in oxygen and fuel consumption. For purposes of comparison, the 1949 design is shown in Fig. 7. A most important corollary advantage of the 1950 design is a proven life expectancy of 3800 ft of hole against 300 ft of hole for the 1949 burners.

The flow of oxygen, fuel, and water is controlled by a lighting sequence switch in the operator's cab, which actuates solenoid valves in the respective lines. The burner is ignited by the machine helper who passes a kerosine flare under the burner while the machine operator performs the proper lighting sequence. After lighting, the helper makes minor flow corrections to maintain established flow conditions. Direct-reading, float-type meters are used.

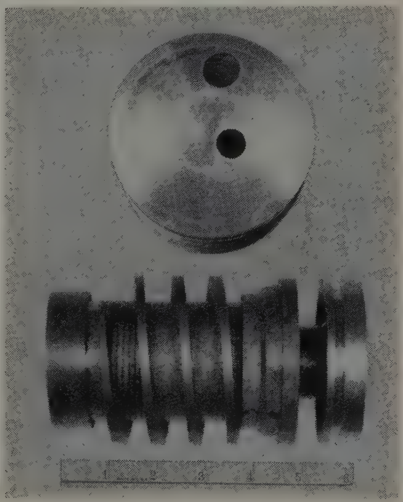

Fig. 7—An earlier design of dual-orifice burner used in 1949.

Process Fluid Supply

In the present operation, oxygen is supplied from a Cascade oxygen unit, as shown in Fig. 8, near the mine buildings. A pipe line runs to the pit where convenient taps are located. The lateral line, consisting of easily handled, 300-ft sections of copper tubing, terminates at a regulator in the pit. Connection to the machine is made by 150 ft of rubber hose from the regulator.

Kerosine or fuel oil is pumped to the blowpipe from a 500-gal storage tank built into the deck of the machine. This supply is adequate for two-shift operation. Fuel is brought to the machine by pipe and hose lines from a central storage unit or by tank truck.

Since it is planned to operate taconic properties throughout the year, a constant water supply is the

Fig. 8—Cascade oxygen unit near the mine buildings.

greatest problem. Temperatures of —40°F are not uncommon in Minnesota. In the present operation, water is brought from the mill reservoir to the pit in a pipe line buried below the frost line in the adjacent overburden. Along the side of the pit are located convenient outlets from which lengths of insulated pipe line, equipped for easy drainage, are laid on the surface to the drilling site. Connection to the machine is made by 150 ft of rubber hose. Water supply presents no problem in the warm season, but in winter weather, freeze-ups in the machine would occur if proper provisions were not taken during overnight or other protracted shutdowns.

In the present setup, the exposed water line is drained at the termination of the working shift, the hose from pipe line to machine is drained and stored inside the machine, which is heated around-the-clock by electric space heaters. The water tank, pump, and lines within the machine are drained. The blowpipe is purged with compressed air supplied by a compressor, part of the equipment of the JPM-1 machine. The burner and hole sizer are removed from the blowpipe and stored in a warm spot in the machine.

In the morning, as soon as the water hose is hooked up, the surge tank located in the rear of the machine is filled. About 5 gal of water are preheated to approximately 125°F in an electric heating unit in the machine. At the same time, the burner and hole sizer are being mounted on the blowpipe. The hot water is passed immediately through the blowpipe, followed by the cold pipe line water. Once flow is etablished, there is no problem of freeze-up. It is never stopped during the day's operation except for hole sizer changes. It is interesting to note that residual water left in the bottom of finished holes never freezes since it is well below the frost line.

Water failure or reduction of water flow before the critical point would destroy the burner; therefore, several safeguards are built into the machine. The water is fed into a 400-gal surge tank, which is kept full by a float valve. The pump drawing from the surge tank delivers 1000 gal per hr to the blowpipe at 80 psi. If the water level in the surge tank begins to drop, a warning light and buzzer in the operator's cab indicate that water adequate for only 20-min burning time remains. The second safeguard is a water-oxygen interlock that automatically shuts off the oxygen if the water flow drops below the minimum safe value for good burner cooling. This interlock, located between the pump and blowpipe, safeguards against pump or piping failure.

Approximately 50 pct of the water discharged in the hole during piercing flashes to steam and, as previously mentioned, it is this steam plus the products of combustion that carry the cuttings continuously from the hole. The steam and burned gases are exhausted from the working area in front of the machine, as shown in Fig. 9, by a 10,000 cfm fan driven by the 40-hp propulsion motor and discharged from a stack to the rear of the machine. The coarse cuttings drop out of suspension onto the ground around the top of the hole and are periodically shoveled out of the way by the helper. Fine cuttings are exhausted through the stack.

In many instances it is necessary to start a hole in broken and rubbly ground rather than in solid rock. In such instances the top of the hole, until solid rock is reached, will enlarge to a cavity that may be 2 ft across. This condition hinders proper discharge of cuttings because the steam loses velocity in this enlarged cavity and can no longer maintain the cuttings in suspension. The cuttings tend to boil around in the cavity and to prevent this the hole is cased to the depth of the cavity and the casing allowed to extend 4 to 6 in. above normal ground level, as shown in Fig. 10.

Piercing Results

Current mining operation is on a 25-ft bench with blastholes being drilled 5 ft below grade. The JPM-1 machine has a capacity for drilling holes of 31-ft maximum depth. In large-scale taconite operations, a higher bench is envisioned, probably around 35 to 45 ft; consequently, future piercing machines will require a higher mast and a longer blowpipe to accommodate this increased height of bench.

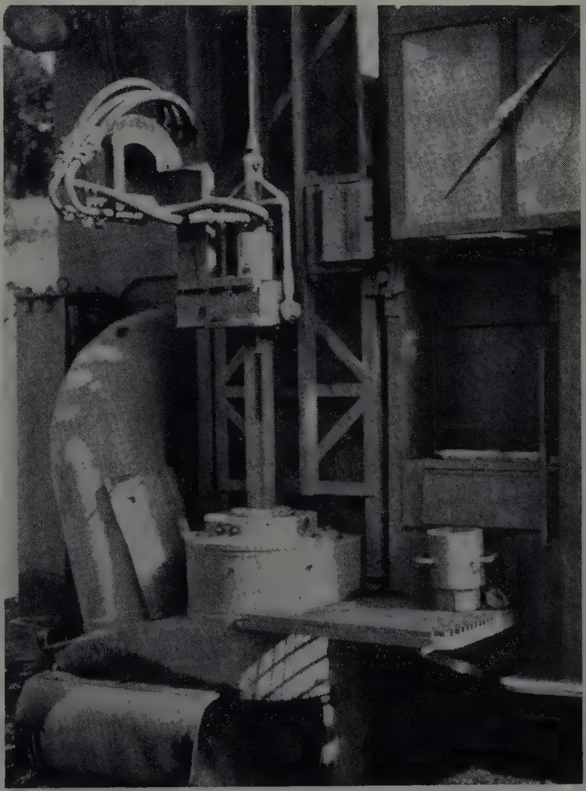

Fig. 9—Steam and burned gases are exhausted from the working area in front of the machine by a 10,000-cu ft per min fan driven by the propulsion motor.

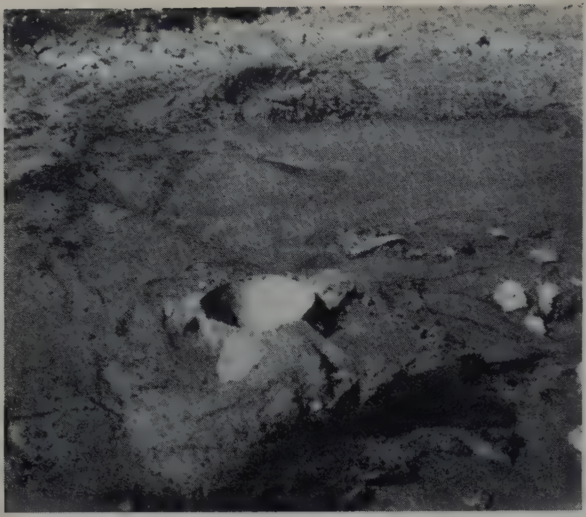

Fig. 10—A hole cased to the depth of the cavity. The casing extends 4 to 6 in. above normal ground level.

Table I shows operating results obtained in 1950. In 1949 for the same oxygen and fuel consumption, the old-style burners would only pierce a 6-in. diam hole, which was loaded with 5½-in. explosives. In 1950, 6½-in. holes were loaded with 6-in. explosives and, a 20 pct increase in ore blasted per hole resulted. This improvement is the direct result of advances in burner design.

Preliminary tests also have been made with promising results on 7½-in. holes for 7-in. explosives. The 7½-in. hole was not used as a regular production item in 1950; however, it will probably be in much greater use during 1951.

Piercing represented by these results was done by regular mine labor, some of whom had been experienced in churn drilling. The technique of jet-piercing was learned easily by these men.

As is to be expected, drilling in solid rock gives the highest process efficiency. As a matter of fact, some holes have been drilled at rates exceeding 30 ft per hr. In bad backbreak, the piercing speeds occasionally drop to 30 pct of those obtained in solid rock.

Where pit conditions permit, it has been found desirable to utilize double-row piercing to bring down a greater tonnage per blast and to realize economies of piercing a second row of holes where the backbreak is minimized. Piercing speeds in the second row have averaged 8 pct higher than speeds in the first row.

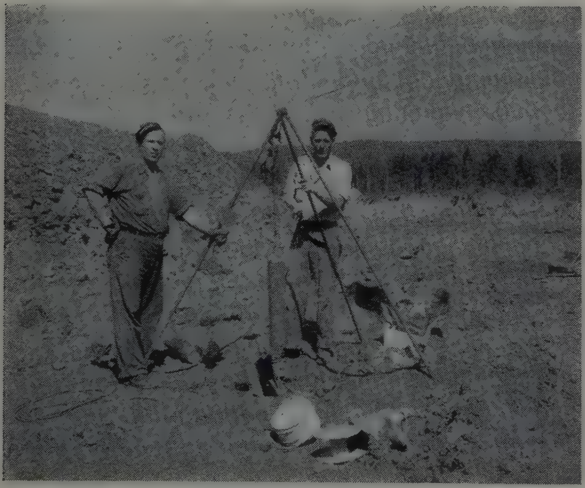

Fig. 11—Explosives being loaded into the hole.

During the summer of 1950, drilling was started for the second lift in the new taconite pit at Aurora. It was necessary to drill through the top 4 or 5 ft of broken rock that lay on some areas of the quarry floor. Piercing through this layer of rubble was about 30 pct slower than when starting on solid ledge. The top few feet of 65 pct of the holes pierced in this area were cased.

It has been found necessary to pierce a hole with a minimum diameter ½ in. greater than the diameter of the explosive to be placed in the hole. Under these conditions explosive loading, as shown in Fig. 11, proceeds without difficulty. Both gelatin dynamite and Nitromon have been used for blasting jet-pierced holes. The load factor, i.e., pound of explosive per cubic yard of burden, used with jet-pierced holes is equivalent to the load factor used with holes produced by other methods of drilling.

A check on 20 holes showed a maximum temperature of 154°F immediately after piercing. This temperature was measured in the vapor zone immediately above the residual water that remains in the hole. The depth of residual water has varied from 6 in. to 8 ft, the latter is the exception. Many of the holes have been loaded within ½ hr after piercing.

Table I. Jet Piercing Operating Results

Avg piercing speed, ft per hr	15
Hole size, minimum in. diam	6½
Explosive cartridge loaded, in. diam	6
Approx. breakage, tons per ft of hole	24½
Approx. oxygen consumed per ft of hole, cu ft	660
Oxygen per ton ore, cu ft	27
Kerosine consumed per ft of hole, lb	18.3

Visual sighting indicates that rather than being smooth, the hole is corrugated. It was desirable to determine the extent of these corrugations and their effect on overall hole volume in order that the possibility of effectively utilizing this extra hole volume might be studied fully. A simple device was made and from the data thus obtained contour measurements were plotted. Illustrated in Fig. 12 are a 6½-in. minimum diam jet-pierced hole and a churn drill hole made with a 9-in. bit. By and large the jet-pierced hole substantially exceeds the minimum diameter set by the reamer teeth. Seventeen 6½-in. holes contoured had an average diameter of 9.3 in. The hole diameter produced is actually a function of the piercing speed: the greater the piercing speed, the lower the average diameter. This is to be expected since there is a relatively fixed relation between the cubic feet of oxygen used to pierce the hole and the cubic inches of rock removed per cubic foot of oxygen. Where the piercing speed is slow because of backbreak and rubble, the average diameter of the hole is consequently larger because piercing time is greater and more oxygen is used.

It is interesting to note that the churn-drill hole is not smooth-sided and has an average diameter of 11.6 in.

Maintenance and Supply

The problem of machine maintenance is one of considerable interest. The JPM-1 jet-piercing machine is the first of its type. It was over-designed to meet as many operating conditions as could be foreseen. While the machine is not perfect, no basic shortcomings have developed. However, excessive wear has been encountered in the exhaust fan scroll casing and impeller, and steps will be taken to overcome this. Kelly wear has also been substantial. The lower end of one kelly has been worn down ½ in. on the diameter in 10,000 ft of piercing. This will be remedied either through building up plus hardfacing procedure or removing the short worn section and welding in a new length. The turntable must be more rugged and larger to accommodate casing pipe. New production machines will have the same components that were included in this first unit. The mast will be higher and will be provided with heavier bracing for rigid hole-to-hole moves. The higher mast probably will dictate a broader machine base to maintain stability. Equipment arrangements will be simplified to permit easier access for maintenance, routine inspection, and lubrication.

The matter of process supplies for full-scale versus pilot plant taconite production is under consideration. Because of the scope of large-scale operation, circulating water pipe lines probably will be mandatory, with provisions for heating the water in cold weather. The main supply line probably will be buried below the frost line and back of the working face. Lateral supply lines that can be drained during periods of no operation will bring water to the rigs. Driox oxygen installations will readily supply the large volumes required and also permit great flexibility in demand loads. Fuel will be brought to the rig either by tank truck or pipe and by hose line direct to the unit.

Cost

In the present small operation, oxygen is much the biggest item of cost. This will change as the mining operation expands. It is common industrial practice for oxygen to be sold on a sliding scale basis—the more used, the lower the unit cost of the oxygen. This factor bears directly on piercing cost, all other things being equal. The bigger the mining operation, the greater the oxygen consumption and consequently the lower the jet-piercing cost. On the present scale of operations, piercing costs per ton are about on a par with churn drilling.

There are certain hidden economies accompanying the use of jet-piercing. A jet-piercing machine at present is doing the job of about eight churn drills. In an area such as northern Minnesota there is insufficient housing for the influx of workers who will be needed for the full-scale taconite plants. Directly or indirectly, the mining companies will have to take care of these needs. At least ten times more housing would be needed for churn drilling than for jet-piercing workers because of the extra help required with bit dressing, trucking, and the like.

New Developments

Certain developments now being tested hold great promise of making previously quoted production figures obsolete. The most important of these from a management standpoint is the development of completely automatic control of the piercing operation. When collars or restrictions are encountered in the hole, the machine senses this automatically, retracts the blowpipe a given distance for a predetermined amount of time, and then readvances past the restricted spot. The machine can be set, therefore, for the most efficient operating condition, and the human factor will be substantially eliminated. The first automatic control arrangement is being tested on the present machine, and substantial improvements over this first arrangement are expected.

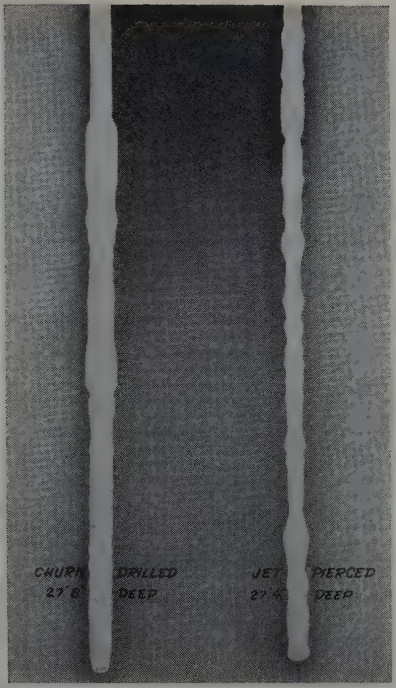

Fig. 12 (Left)—A churn-drilled hole made with a 9-in. bit. (Right)—A jet-pierced hole made with a 6½-in. hole sizer.

An important feature from the process standpoint is a technique known as hole enlargement, which allows the piercing of a wide range of hole diameters with one machine, changing only the burner and hole sizer. This technique requires first that a pilot hole be drilled to the required depth. Referring back to the hole contour chart, it will be noted that the amount of material to be removed in increasing the diameter of hole 1 or 2 in. over its minimum diameter is quite small. For instance, with a 6½-in. diam pilot hole, the amount of material to be removed to make a minimum 7½-in. hole is only about 3 pct of the original hole volume. To make an 8½-in. hole, the volume to be removed is 6 to 8 pct. A special radial-type flame burner, with a hole sizer for the large diameter hole, is mounted on the blowpipe for enlarging a pilot hole. The blowpipe is dropped at maximum speed between restrictions and slowed down to cut through the restrictions. Speeds on this enlarging pass in making a 7½-in. from a 6½-in. hole have been as high as 69 ft per hr. The 6½-in. diam holes have already been enlarged to as great as 9¾-in. diam. Studies are being made to determine the optimum pilot-hole diameter in relation to the final enlarged diameter. The possibilities of such a technique provide a wide degree of flexibility in hole diameters in one piercing machine.

The drilling of blastholes has been one of the problems confronting the commercial development of magnetic taconites. There are strong indications that jet-piercing has met and overcome this problem by giving a high unit production per machine at a reasonable cost per ton of ore in front of the shovel. To these advantages are added the flexibility possible in hole diameters and the current development of automatic control in jet-piercing.

Recent Operating Improvements At Kennecott's Utah Copper Mine

by L. F. Pett

ALTHOUGH Kennecott's orebody has long been outlined, it is still necessary to define further its limits. This mine, long an advocate of churn drill methods, recently supplemented its practice by using a diamond drill. So far, the diamond drill holes have been along the main canyon floor which traverses the mine area. This zone, highly fractured, is cut by numerous friable quartz veinlets encountered within the porphyry. These veinlets, formed by filling contracted porphyry fractures with hot, siliceous, mineral-bearing solutions, are often more highly mineralized than adjacent porphyry. They pulverize on contact, yielding only slight amounts of core.

For this reason neither analysis of core recovered (amounting to 37.5 pct of approximately 10,000 ft of hole drilled) nor sludge analysis considered separately would give correct assays. The following equation has been adopted:

$$\frac{\text{actual weight of core} \times \text{core assay} - \text{theoretical weight sludge} \times \text{sludge assay}}{\text{actual weight core} - \text{theoretical weight sludge}} = \text{assay}$$

Drilling is done under contract. Collection of core and sludge samples together with double-checked assaying is done by the company. Microscopic, spectographic, and X-ray examination of rock specimen saved is carried on at the University of Utah. A fund, set up by Kennecott Copper Corp. at the University, has encouraged cooperation in solution of mining research problems.

The drill used is a Longyear hydraulic powered by a Waukesha 26-hp gasoline motor. A pump, Bean Royal type, is powered by a 7-hp Briggs-Stratton gasoline motor. Weight of unit is 4000 lb and is placed under tripod of 35-ft wood poles within a 25-ft sq area.

The following results have been attained:

		Core Size, In.
Avg depth of hole drilled, ft	1215	
Avg drilling depth per 8-hr shift, ft	10.11	
Avg depth of NC casing, ft	21	2 11/16
Avg depth of NX casing and size core, ft	178	2⅛
Avg depth of BX casing and size core, ft	427	1⅝
Avg depth of AX casing and size core, ft	585	1 5/32

Through the use of metal sludge collection boxes, 12 ft long, 18 in. wide, and varying from 12 to 24 in. in depth, sludge recovery has been maintained above 95 pct. Samples, both core and sludge, are assembled for each sample interval and averaged in 50-ft bench increments. Determinations for mineral values are made at both the mine and mill. Assay results varying in excess of 0.05 of copper assay values are rerun to determine correct assay.

The organization chart for the Kennecott Utah copper mine is shown in Fig. 1. The Utah mine requires approximately four men per shovel shift to drill solid rock, properly slope banks, and trim loose material from the top of bench sections in making them safe for men and equipment working below.

During recent years, while the mine has been worked day and night, powder crews have worked day shift only. Series of toe holes drilled back of advancing electric shovels comprise approximately 70 pct of all mine drilling. These holes, spaced about 21 ft apart, have a depth of 24 ft. Of the footage drilled, 25 pct is for vertical holes either 24 or 28 ft deep, spaced about 28 ft centers along the top rim of banks. This combination of toe holes, using an Ingersol-Rand DA 35 drifter mounted on a tripod and the vertical holes using an Ingersol-Rand Type X71 wagon drill is standard practice above the pit elevation. These benches are 68 ft high, with a few exceptions, while benches below the pit elevation are 50 ft high and require very little drilling other than toe holes. Five benches, varying in height from 80 to 100 ft, require a combination of toe holes plus vertical holes and bank holes or electric churn drill holes. Bank holes using the DA 35 drifter and electric churn drill using Bucyrus-Erie Model 29T, account for 5 pct of feet drilled. Until the past year, drilling of fill material on upper levels was accomplished by churn drill method. Wagon drills are now used for fills because they can be set up and drill a hole an hour.

It is necessary to run an air drill 7 hr for each 8-hr shovel shift. Average drilling rates during the past year were as follows:

DA 35 drifter using 2 and 3-man crews	— 4.36 ft per man hr
X 71 wagon drill using 2-man crews	— 6.45 ft per man hr
DA 35 drifter bank holes, 3-man crews	— 2.53 ft per man hr
Bucyrus-Erie churn drill, 2-man crews	— 1.9 ft per man hr
Total mine avg	— 4.57 ft per man hr

Mine practice assigns powdermen to job locations where they perform all drilling and blasting work within the area. Almost as much labor is consumed in springing and loading holes as in drilling them. Average powder consumption is approximately as follows:

	Per Electric Shovel Shift, Lb
To spring and chamber holes	— 50
Blasting	— 500
Trim banks	— 50

One fifth of all powder used is Hercules hercomite bag powder while four fifths is Hercules gelamite 65 pct strength in 1¼ in. x 8 in. sticks. Frozen material, overhanging from top banks, creates a winter hazard to shovel operators and equipment. Experiments are being conducted with a high-speed Hercules powder in an effort to eliminate this condition.

Banks are trimmed by powdermen who lodge 8 or 10 bundles of powder, containing 15 or 20 sticks each, across the upper surfaces of each working face. To each bundle of powder is attached a primer with 6-ft fuse which is lighted as the powdermen, who have lowered themselves down the bank on ropes, ascend to safety.

Three angle dozers, D8 caterpillars, make grades and move supplies for the average 31 drilling machines working daily.

Compressed air is furnished by three Ingersol-Rand compressors, each with rated capacity of 3750 cu ft of air per min at 100 lb reservoir pressure.

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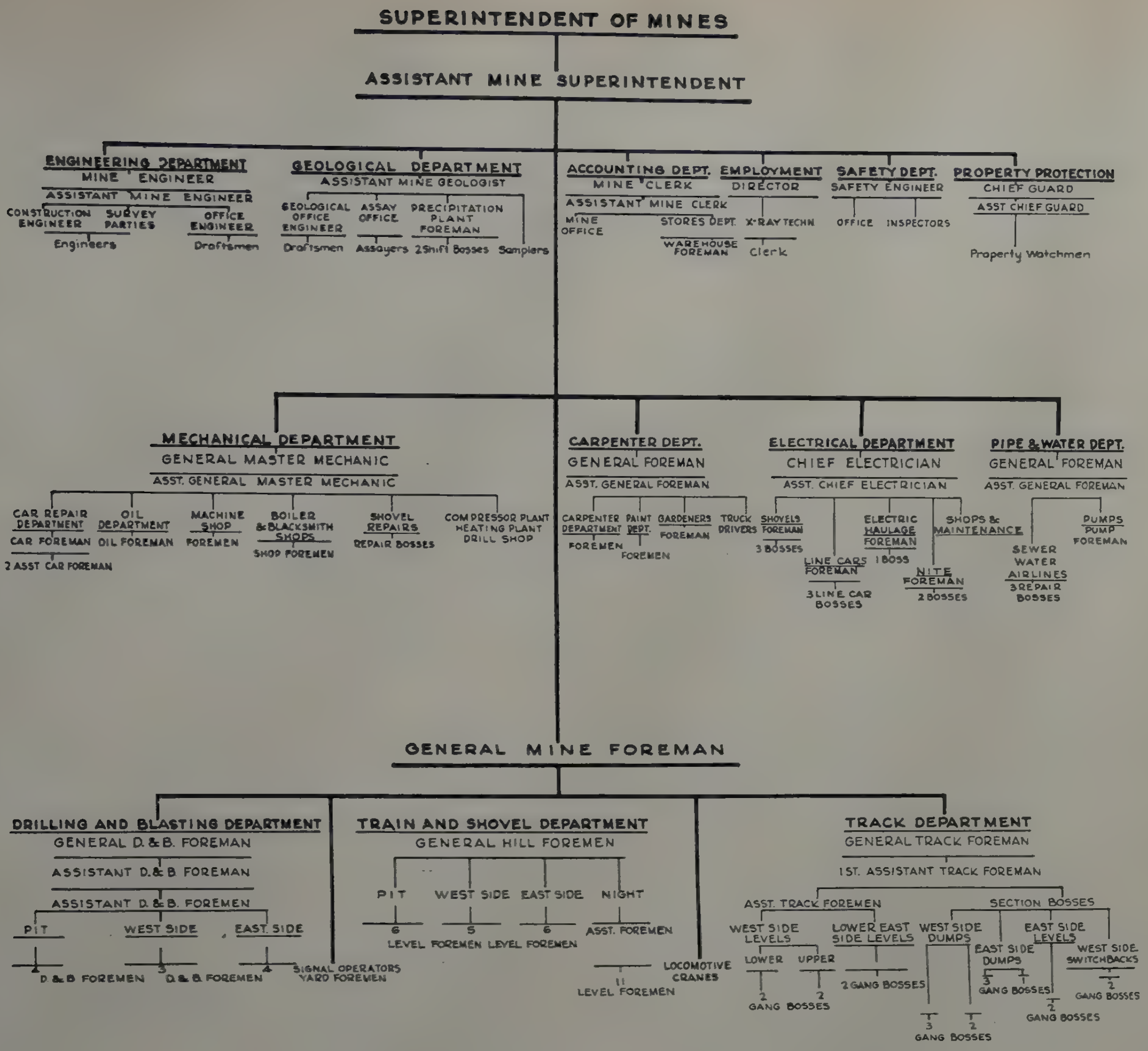

Fig. 1—Organization chart for the Kennecott Utah copper mine.

Numerous minute leaks encountered in normal distribution of the 65 miles of air line within the mine area cause excessive air loss, and unit drilling equipment and supplementary air supply are being investigated.

Track

Into the mine's labor pool come local high school boys, Mexicans, Japanese, Puerto Ricans and Navajo Indians, and from this pool emerges the mine's track department. After the men obtain a few weeks experience, they begin to leave the track for higher paid jobs via a seniority bidding system. Unskilled labor and increasing costs have forced machines to usurp many tasks previously done by well-trained track gangs.

Table I. Increase in Operating Efficiencies

	1937	1940	1948
Ore and waste loaded, tons	51,427,000	56,838,000	53,300,000[a]
Mine stripping ratio	1.223	1.19	1.369
Track lined per trackman shift, linear ft.	30.5	35.3	42.7
Waste dumped per trackman shift, cu yd	160	287	449
Wage rate per trackman shift, $	4.30	4.30	8.89 (6 months) 9.85 (6 months)

[a] First 9 months of 1948.

The Utah mine has not yet found a way of replacing track ties, maintaining track gage or superelevation by machine methods, but its track department has undergone considerable mechanization. Track department workmen are divided into the following three groups: men in gangs, 60 pct of the total; switchtenders, flagmen and yard cleaners, 30 pct; and operators of track machines, 10 pct.

Work assigned to the track department requires approximately 17 men per shovel shift. Among switchtenders and flagmen are many employees who, after long company service, have returned to these ranks. Extension of centralized traffic control on ore haulage switchbacks will remove a few of these men by pushing younger and more able-bodied men back to gang labor.

The company's program of replacing 65-lb steel on level bench tracks with 90-lb steel and changing 90-lb steel to 131 and 132 lb on ore haulage switchback is nearing completion. To date, 45 miles of track have been replaced by heavier steel. This program, initiated because of increased tonnage of ore and waste cars, also permits higher speed traffic to match longer ore and waste haul.

The years 1937, 1940, and 1948 are comparable in mine output and offer comparison in track mechan-

Table II. Hoist Cables Used

Number of Hoist Cables	Construction a[a]	b[b]	Cost Per Ft	Cu Yds Loaded, Avg
63	6 x 19		0.818	164,833
62	6 x 29		1.003	233,101
64	6 x 37		1.032	243,994
407	6 x 41		1.083	256,136

[a] a—Strands.
[b] b—Wires per strand.

ization effected in dump maintenance, necessary for waste disposal, and lining bench line track in shovel pits. The 99,000 track shifts required during 1937 were reduced to 68,000 shifts by 1940 and further reduced to 46,000 shifts during 1948. Table I shows progress made in these years.

The period from 1937 to 1940 reflects introduction to full coverage of angle dozers for level grade preparation as well as filling and grading of dumps. D8 caterpillars using 11-ft blades were adopted for bench lines while D7 caterpillars using 10-ft blades were originally used on dumps because caterpillar treads corresponded to track gage. D7 caterpillars were replaced with D8 machines on dumps. The more recent progress was the result of the introduction of miscellaneous equipment. A caterpillar, diesel No. 12 motor grader, increased feet of track lined per track shift by 27 pct during the past year. This machine adjusts track superelevation to curvature and practically eliminates raising track for grade adjustments. Jackson electric tie tamper units, Model M2, with a 4 to 6 hp gasoline motor driving a 2.5-kw generator, delivers 60-cycle, 110-v ac to vibrating tampers with 6-in. blades. These units boosted feet of track raised and tamped per track shift 31 pct and improved track surface. Spike pullers, spike drivers, pavement breakers, power track wrench, rail grinder, and welder are used by mine track crews.

Shovels

Since tonnage per shovel shift signalizes the degree of success in open-cut mining, more shovel power without sacrifice of mobility was sought. The Marion full-revolving shovel, Type 4161, with 5-cu yd dipper, 1⅜-in. hoist cable carrying 100,000-lb bail pull, in conjunction with 40-cu yd waste cars, 100-ton ore cars and 85-ton dc locomotives is the Utah mine's nucleus of production. Three-man shovel crews and two-man train crews are used.

Six shovels have been purchased with dual operating compartments. These compartments increase maintenance costs but were adopted because they increase the safety of operating crews.

Amplidyne control systems have been furnished as standard equipment on four shovels. No appreciable increase in loading has been made by these shovels. A slight operators' resistance to shovel kick-out on stall or overload exists. Excessive heat developed in the grid control panel, as a result of carrying higher and more uniform sustained voltage than Ward Leonard system, will probably require an additional exhaust fan on the roof immediately above the grid panel. Difference in size and use of electrical contacts reduces electrical maintenance work. These shovel controls have not been used long enough for conclusive mechanical maintenance comparison but indications point to a substantial reduction in mechanical maintenance on Amplidyne shovels.

One shovel house has been rebuilt in an effort to add rigidity to house protection. Five-in. H-beams, spaced 4-ft centers along sides and arched top, support 5/16-in. plate on house covering and ½-in. plate on operators compartment. Cost of this change is approximately $5000 per shovel. Three serious accidents focused attention toward additional house support. It is anticipated that such shovel housing will be adopted in future construction with present houses converted as rapidly as mine crews can make the change.

Replacement of shovel parts most frequently worn includes hoist cables, dipper sticks, dippers, latch plates, and dipper teeth. Experiments are being conducted on 6x46 hoist cables but results are not yet conclusive.

Dipper sticks with inside handle used on Type 4161 shovels show wide variation in yardage loaded, but an average of 1,343,000 cu yd per handle was loaded during the past five years. Most common point of failure is midway along the handle which may be caused by the mine's practice of starting swing movement as soon as the hoisting operation is in full motion.

Except for three dippers of the Marion type, American Manganese 5-cu yd dippers are standard on revolving shovels. Dipper changes, which are listed in terms of change of dipper front, have been made after an average loading of 517,000 cu yd. Approximately four latch plates are used for each change of dipper front. A set of dipper teeth is required for approximately each 100,000 cu yd of ore or waste mined.

Ore at depth in relation to remaining ore above the pit elevation makes mandatory increased speed in carving 50-ft benches below present working levels. During recent years these levels have been established in about 18-month intervals. A Marion shovel, Type 4161, equipped with 35-ft boom and 5-cu yd, short bail dipper has been used to dig a shelf 6 ft deep. Successive drop cuts follow each track line, toe hole blast, and electrification cycle. Past practice has been to form, roughly, a circular loop 800 to 1000 ft in diam before beginning a new level with its 1800 ft of 3 pct switchback and 1000 ft of level room.

Last year, in an effort to increase progress, the bottom ore shovel was placed on a three-shift basis with shovels on adjacent levels working two shifts daily. To further augment this program, the company is purchasing a Marion 45-ft boom with 32-ft dipper sticks and 4-cu yd dipper, which will replace a standard boom and accomplish the 50-ft drop in a series of three cuts rather than the present nine. Wagon drills with vertical hole drilling will replace the present toe hole method of shooting solid material.

Electric power comes from the company's 100,000-kw steam plant 15 miles distant. Power at 42,500 v, three phase, 60 cycle, is received over two circuits of No. 4/0 bare copper conductors supported on steel towers. Copperweld static wires are used to protect these lines from lightning. Four substations with transformer capacity of 26,750 kw are strategically located for power distribution. Substations No. 1 and No. 2 are located on opposite rims of the mine working benches. Each station has two banks of three phase, 1200-kw Maloney transformers connected in parallel. Connections are delta-delta 40,000 to 5500 v. From each station emerge four, three-phase, 5500-v ac feeder circuits stretching finger-like to touch opposite ends of each mine level in supplying power to 42 electric shovels. Bench lines traverse each level, end to end. Power lines leading to shovels are three phase, No. 2, bare copper wire carried on movable steel towers. Lines are energized only to point of contact with the shovel's trail cable. As the 5000-v power lines leave feeder taps to enter the level bench lines, they pass through a 7500-v, 600-amp, three-pole, oil-circuit breaker having an overcurrent trip attachment. These switches are manually operated. In leaving substations, feeder lines pass through a General Electric oil circuit breaker of 15,000 v with a 400-amp rating. These breakers have three-pole automatic overload tripping which is controlled by General Electric Type 1 AC induction time overcurrent relays. Breakers trip instantaneously on short circuit, and in 3 sec at 650 amp for sustained overload. Being closer to the source of power and more sensitive than bench line equipment, station breakers normally trip ahead of bench line circuit breakers, unfortunately putting out of service a group of shovels.

Rotary Converters

Mine haulage electrification was established on a basis of 750-v dc. This current is supplied from four substations spaced to meet feeder requirements. Stations 1, 2, and 3 use synchronous rotary converters. These machines, ten in all, are Westinghouse converters of 1000 kw 750-v six phase 900 rpm with overload rating of 150 pct for 2 hr and 200 pct for 30 min. All converter stations have automatic switching and control equipment. However, a station attendant is maintained on day shift. An Exide battery, Type EMGO, 7-plate, 120-amp hr capacity with 120 v, is used for control energy. A Westinghouse M.G. set, 440 v, 3-phase, 5½-hp motor with generator rated 3½ kw on 125 v, is used to charge the station battery. Direct current from each substation is delivered over million circular mil feeder lines to the various trolley circuits. A circuit breaker, Westinghouse, Type CH, 2000-amp, 750-v, air break equipped with automatic short circuit detecting resistance is placed in each feeder circuit.

Because of excessive heat liberated by converters, it was found necessary to install a forced ventilation system in No. 3 station during the past year. Air is drawn through filter by fan, which forces air through ducts leading to each machine. Five or six air changes are made per hour. Heated air escapes through ventilators on the roof. This circulating air dropped temperatures 8° to 10° during warm weather.

Mercury Arc Rectifiers

Substation No. 4, placed in service during the past year, provides power for ore haulage to the newly electrified main line assembly yard. It is equipped with two 750-kw mercury arc rectifiers in place of synchronous rotary converters. The Ignitron rectifiers are six tube, Westinghouse Type 1 P.G. 36, 750 v. These machines do not require a station attendant, but equipment receives daily inspection. The only difficulty experienced has been caused by occasional power interruptions. Instead of waiting for the temperature to reduce and permit automatic closing of switches, an electrician closes them manually. Pressures within tubes rise rapidly when power interruption occurs. Reducing this pressure by vacuum pump normally requires 10 to 15 min. These occasional interruptions are bridged by a million circular mil tie between the No. 3 substation, with rotary converters, and switchback and yard normally supplied from the arc rectifier substation. Absence of moving parts, elimination of daily station attendant and flexibility of overload amounting to 150 pct for 2 hr and 300 pct for 5 min have proved these machines well adapted to mine service.

Mine Haulage

Sixty-seven 85-ton General Electric locomotives, having a 250-hp series motor mounted to each of four driving axles, are used in mine haulage. These locomotives have a tractive effort of 42,000 lb when trolley carries 750 v. Drop in voltage brings a corresponding reduction in available effort. Maximum tractive effort is based on a speed of 10 miles per hr. Each locomotive is equipped with a compressor capable of furnishing 150 cu ft per min at 130-lb reservoir pressure providing ample air for the average 7.3-mile ore haul over grades varying from 3 to 4 pct. Gear ratio between motor pinion and shaft gear is 85 to 18 or 4.72 except on one locomotive which has a gear ratio of 76 or 4.47. Each locomotive carries side arm collectors with height range from 18 to 20 ft which are used on work levels. Overhead pantographs having effective height variation from 18 to 24 ft are used on switchbacks and tracks beyond working levels. Empty ore trains, consisting of a locomotive and 16 ore cars, travel up 3 and 4 pct switchbacks while loaded trains have from 15 to 22 cars on return trip down grade. Empty ore cars, weighing 25 tons, are loaded with 91 tons of ore. Speeds vary from 8 to 10 miles per hr up grade or over working levels and from 10 to 20 miles down grade. Because of the increased length of haul for ore trains and resultant over-heating of electrical equipment, ventilating blowers are being installed on locomotives in this service. Two 3½-hp blower motors, one for each truck, will draw air through filters and deliver 3000 cu ft, which will be divided between two drive motors and the enclosed grid compartment. Increase in width of motor pinion and gear teeth is being considered, as is an increase of motor drive shaft from 4 in. to 4½ in., in an effort to reduce shaft deflection, a frequent cause of gear failure.

Waste Cars

While both 30-cu yd and 40-cu yd waste cars are in service, the company's present program is expected to provide ample 40-yd cars within two years. Waste trains, consisting of locomotive and seven waste cars travel over switchbacks varying from 3 to 5 pct. Locomotive compressor provides air for brakes and dumping cars as well as furnishing air for a light generator. The waste trains are loaded by shovels and then pushed by the locomotive towards the dump. At night the locomotive headlight is not visible over top of loaded cars, so that additional light must be provided along track ahead of moving cars. Air generators, Ingersol-Rand, size AL 150, are used in delivering 115 v to a 150-w lamp inside seal beam container. This lamp, hanging from the front end platform, furnishes sufficient illumination for a distance of 250 ft.

Felpax journal lubrication has recently been installed on ore cars and 40-yd waste cars. This lubrication improves car journal wear and practically eliminates cold weather hot boxes.

A G.R.S.* centralized traffic control system has been incorporated into the mine's ore haulage switchback system. Information concerning position of switches and location of trains is provided by light indicators on two control panels. The lower control tower is located in the main assembly yard at Copperton, five miles north of mine terraces. From this tower main line trains with double unit, 250-ton electric locomotives handling 70 to 90 loaded ore cars are directed to either of two mill yards 12 miles away. Empty trains of equal length return from the mills, following centralized traffic control signals to Copperton yard. As mine locomotives leave this yard, with 16 empty ore cars, they enter mine switchback tracks under control of the central yard tower, situated midway between the mine and assembly yard. From this point empty trains fan out to various mine entrance levels. Loaded ore trains are likewise collected, and traffic between central yard and Copperton is over a double track system. Model 5B switch machines with 20 dc v rating are used to throw switches. Included in the present centralized traffic control system are 66 signals and 27 switches. Switch point heaters, using 220 v ac power, are installed with each switch. Track signal circuits are controlled by 8 v dc. Centralized traffic control equipment has speeded up traffic safely and eliminated 16 switchtenders and flagmen on mine tracks.

To provide immediate contact with electrical and mechanical maintenance crews, as well as operating personnel, 13 radio installations were made at the mine during 1948. Equipment includes four land stations and nine mobile units and was installed in connection with the Mountain States Telephone Co. With the exception of a 50-w mine office station, which receives its power from 110-v ac lighting circuit rectified, all stations are of 30-w capacity and receive power from a 6-v 160-amp hr storage battery. This power drives a motor generator set producing high frequency modulation necessary to transmit and receive voice messages. All stations operate on a single frequency to transmit or receive. Push button control, on handle of hand set, shifts station equipment to receive or convey signals. While the carrier frequency used is known as "line of sight radiation," messages carry over curvatures well beyond line of sight but quickly fade when an abrupt ridge separates stations. The four land stations can, by single relay, reach any point of the 150 miles of track. Mobile radio should tend to compress the mines sprawling operations and place them firmly in the hands of supervision.

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[*] General Railway Signal Co.

Technical Note

Flotation of Diamonds

by R. G. Weavind, I. Wolf, and R. S. Young

ONE of the most important fields of investigation at the Diamond Research Laboratory in Johannesburg is concerned with improvements in metallurgical practices for the diamond mining companies, with particular reference to increased recoveries of the smaller industrial stones. Considerable work has been done on the possible application of flotation processes to the diamond industry.

From the viewpoint of recovery processes, diamonds in their natural state may be grouped into two classes, the water-repellent diamonds, which adhere to a grease surface, and the water-avid diamonds, which, because of their wettability, will not adhere to a grease surface and must be recovered by means other than the grease tables used for water-repellent diamonds.

Experiments in the flotation of small diamonds from both kimberlite and alluvial deposits indicate the following conclusions:

Water-repellent diamonds: Satisfactory flotation of —16 mesh diamonds, which are naturally water-repellent, may be accomplished using either Aerofloat 25 together with cresylic acid, or Du Pont B23 as frother together with kerosine as auxiliary oily conditioner or froth stiffener. The quantities of reagents required are of the order of 0.15 lb per ton Aerofloat and 0.18 lb per ton cresylic acid.

The use of acid circuits is detrimental, and for best results a pH between 7 and 9 should be maintained. A flotation period of 5 to 10 min is necessary to insure that all floatable diamonds are removed.

Diamonds larger than 16 mesh cannot be recovered satisfactorily by flotation.

Xanthates do not act as collectors for diamonds, and this is probably associated with their inability to impart a water-repellent surface to this mineral.

Water-repellent diamonds smaller than 28 mesh can be floated quite well by the addition of a frother alone, such as pine oil or Du Pont B23.

Water-avid diamonds: Partial flotation of the smaller sizes of wettable diamonds can be achieved by preconditioning with fatty acid soaps, cationic amines, and similar reagents, which will impart a water-repellent film to diamonds. Washing to remove excess reagent, followed by flotation, will result in a fair recovery, but the latter is much lower than for water-repellent diamonds.

Crushed boart: Recovery of crushed boart from the waste products of industrial uses is best accomplished by first conditioning the diamonds present in the mixture with fatty acids, cationic amines, etc., removing the conditioning agent by washing, and floating in a cell of the pneumatic type. Flotation reagents found to be most suitable are Du Pont B23, and Aerofloat 25 together with cresylic acid. The optimum quantities are 1.2 g Du Pont B23, and 0.55 g Aerofloat plus 1.4 g cresylic acid, per liter of pulp in the cell. A solids-to-water ratio of 1 to 4 is found to be the most suitable, and alkaline circuits are necessary for best results. Diamonds larger than 28 mesh are not recovered satisfactorily by this procedure, but for finer sizes flotation offers an excellent procedure for the reclamation of this strategic industrial material.

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Economic Significance Of Recent Technologic Research On Solid Fuels

by Arno C. Fieldner and Ralph L. Brown

PROBABLY no large industry in the United States is in greater need of technologic research leading to economic improvement than the coal industry. It has suffered severely from increasing substitution of petroleum and natural gas for coal and its products in the fuel markets. Part of this substitution is due to the greater convenience and suitability of liquid and gaseous fuels for special purposes, and part of it has been due to the extensive research programs conducted or sponsored by the petroleum and allied industries in the past 2 decades. A recent estimate[1] places these expenditures at $100,000,000 a year.

Expenditures on coal research and engineering development probably approximate $20,000,000 a year. The Government is expending about 25 pct of this amount on the production of synthetic liquid fuels from coal. Excluding synthetic liquid fuels, the total expenditures on coal research and development by all agencies, public and private, may be estimated at about $15,000,000. This is only 0.5 pct of the value of the coal production in 1947—688 million tons. This is a modest figure in comparison to the $100,000,000 a year spent by the petroleum industry; its expenditures are about 3 pct of the value of the production of petroleum for the same year.

From a superficial point of view, this comparison appears unfavorable to the coal industry. However, it must be remembered that even today only 15 to 20 pct of the coal production is processed in the manufacture of gas, coke, and chemical products. Eighty to eighty-five percent is burned raw or washed to reduce its content of sulphur and ash-forming minerals, whereas all crude petroleum had to be subjected to distillation or other chemical processing before it could be used. Such processing required the services of chemists and research engineers to make petroleum useful as a fuel or a lubricant. Research thus became a recognized essential for securing and increasing the markets for petroleum products, and naturally the idea of scientific research grew up with the industry and spread to exploration and production as well.

The principal problems of the coal industry were in mining the coal from the bed and handling it at low cost. Research on better utilization was left to the users—power-plant-equipment builders and operators, the coke and gas-manufacturing industries, and government and educational research agencies. Very few coal companies undertook research, unless they may have cooperated with manufacturers in developing better mining and preparation equipment.

This condition began to improve in 1930 when the anthracite-mining industry of Pennsylvania established a research laboratory to improve anthracite-burning equipment for residential and industrial heating. In 1933, the bituminous-coal industry likewise undertook to support bituminous-coal utilization research by establishing Bituminous Coal Research, Inc. Beginning with a small appropriation contributed by coal-mining and railroad companies and the Battelle Memorial Institute, the annual appropriation sponsored by Bituminous Coal Research, Inc. has grown to over a million dollars a year. It supports extensive utilization research at Battelle Memorial Institute, important fundamental research at The Coal Research Laboratory of the Carnegie Institute of Technology and coal research by other agencies. Through its Locomotive Development Committee it supports pioneering research on the development of a coal-burning gas turbine and through the Mining Development Committee it promotes research on a new type of continuous mining machine that operates without using explosives.

Other organizations have contributed major publications on fundamental and applied coal research in the last two decades, including the Minerals Industries Experiment Station of Pennsylvania State College, the Illinois Geological Survey and University of Illinois, the University of West Virginia, the University of Michigan, the University of Alabama, the University of North Dakota, the Massachusetts Institute of Technology, the Koppers Company, the U. S. Bureau of Mines, and, in recent years, the Institute of Gas Technology.

Engineering development work, which is included in the overall estimate of $20,000,000, is conducted largely by manufacturers of equipment, and this portion of the estimate is difficult to gage. It represents a large proportion of the estimate and probably is too small rather than too large. For example, the development of mobile coal-loading machines and continuous mining machines, outstanding in recent mining research, undoubtedly involved a high cost.

Coal Mining

From an economic point of view, the greatest research development in the coal industry obviously has been the growing degree of mechanization in mining. The percentage of underground-mined bituminous-coal output mechanically loaded has risen from 3 pct in 1926 to 67 pct in 1949. Coal operators and equipment manufacturers have united in designing machinery. Loading machines of large capacity and astonishing flexibility are in common use. Shaking conveyors transport coal from producing faces to transfer points from which it may be carried long distances underground and on the surface; and in many mines rubber-tired shuttle cars carry the coal from the loader to the

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train of mine cars or belts. The mechanical loader fills a shuttle car in less than a minute, and the shuttle car discharges its load into the mine car in 30 sec. Similar improvements have been made in drilling and blasting.

In the last few years the greatest achievement of all is being successfully demonstrated in the continuous mechanical mining machine that cuts the coal from the face without drilling or blasting. Under favorable conditions, this machine has increased the tonnage of coal mined per man employed at the face, 50 to 150 pct. An important new auxiliary being tried with the new Bituminous Coal Research mining machine is an extensible stainless-steel conveying belt using a conventional shaker-conveyor drive. A radically different type of continuous mining machine is the coal planer developed in Germany during the war. Coal is cut from the longwall face by a plane or plough drawn by a cable. The coal falls on a conveyor parallel to the course of the plough. It works best with the more friable coals, and the Bureau of Mines will test one in this country on thin beds in the low-volatile field of West Virginia in cooperation with industry.

From the point of view of improving safety in coal mining, an outstanding recent development has been the use of roof bolts to prevent falls of roof. Such falls have been for years the greatest single cause of deaths and injuries in mining coal. Roof bolting is accomplished by drilling holes into the roof and inserting mild-steel bolts about 1 in. in diam in the holes. The bolts are threaded at one end and slit at the other. They are anchored at the back end of the hole by driving the slit end of the rod over a steel wedge, which expands the end of the bolt in the hole. The bolts then are placed under tension by tightening the nuts on the threaded end on a bearing plate at the collar of the hole with a power wrench. The effect of roof bolting is to consolidate several thin strata to form a thicker and more competent beam across a mine opening than the thin individual stratum and thus prevent falls of roof.

During the past 2 years the Bureau of Mines, in cooperation with coal operators, has conducted extensive research on this method, and its use has spread rapidly. Practical experience has shown that roof bolting also improves operating efficiency at the face to a considerable degree and, therefore, may be regarded as an important factor in economic improvement of mining operations.

Preparation of Coal

The increased mechanization of mining has imposed additional burdens on the preparation of coal. Mechanical mining necessarily includes more ash-forming impurities in the coal than hand mining with good face preparation. However, these impurities are removed easily and more cheaply by mechanical cleaning plants on the surface. Coal washing also has become more essential because the best grades of coal are being rapidly depleted. In the last two decades the percentage of bituminous coal mechanically cleaned has increased from 8 pct in 1930 to 35 pct in 1949.

Probably the most important technologic development in coal cleaning in recent years is the heavy-medium process of washing, using a semistable suspension of finely ground solids as a flotation bath to separate good coal from refuse. The most popular type of suspension in use in the United States is finely ground magnetite in water. It has the advantage that the medium can be reclaimed and reconditioned with magnetic separators. A strictly American development in this method is the drum-type separating vessel, which is especially adapted to the treatment of large sizes of coal at high tonnage rates.

A relatively new special development of the heavy-medium process is the cyclone washer, a heavy-medium washing vessel in the form of a hydraulic cyclone. It was developed at the Dutch State Mines and is now in use in a few preparation plants in Europe. This cyclone is especially adapted to handle fine sizes of coal efficiently and is reported to be effective down to a size of ½ mm. The hydraulic cyclone is also being introduced widely as a water-clarifying and sludge-recovery device. For this purpose, hydraulic cyclones are fed with the washery water pumped into the cyclone at about 1 to 2 atm pressure. The size of primary cyclone units used for this work has been fairly well standardized at 14 in. diam, and such a machine will handle washery water at the rate of about 250 gpm. Larger cyclones may be used as classifiers to discharge the fine slimes in the effluent and recover the deslimed coarser sludge as a spigot product.

Smaller cyclones, down to as small as 3 in. diam, have been introduced to recover the very fine slimes. Some combination plants have been introduced in the past year to make a complete recovery of washery water, in which two sets of cyclones are used—the first stage consisting of 14-in. cyclones and the second of banks of 3-in. cyclones on the effluent water from the 14-in. machines. A relatively large installation of the Carnegie-Illinois Steel Corp. is operating successfully with the washery water in 100 pct closed circuit.

Another recent development of promising future importance is the kerosine flocculation method of cleaning fine slurries and dewatering the cleaned product. This method is an adaptation of the conventional froth-flotation process, using kerosine as the principal reagent so as to obtain a light froth that breaks easily and permits ready draining of the product. The system developed at the Southern Experiment Station of the Bureau of Mines at Tuscaloosa, Ala., now in commercial use at one of the washeries of Sloss-Sheffield Steel Co., uses a modified screw conveyor to clean and partly dewater the washery sludge at that plant.

In the preparation of subbituminous coal and lignite, significant progress has been made in research on the flash-drying of the fine coal in the entrained or fluidized state. Pilot-plant tests have shown reductions of bed-moisture content of lignite from 60 to 7 pct and less.[2] This is an important step in upgrading these low-rank coals and will reduce the cost of transportation by removing a large weight of water, which is of no use to the consumer.

Combustion of Coal

As research in the mining and preparation of coal has led to the most significant economic results in the production of coal, so has research in combustion, carbonization, and gasification of coal produced continually increasing economies in its utilization. Since about 85 pct of coal production is burned to produce heat or power, it is not surprising that the widespread research in combustion and fuel utilization of coal initiated in the early days of the present century resulted in material economic returns to American industry. The great reduction from an average consumption of 3 lb of coal for the generation of 1 kw-hr of electrical energy by public utility power plants in 1920 to 1.34 lb in 1940 indicates the magnitude of these fuel savings. The cream has been skimmed from the possible gains that can yet be made in a modern public utility power plant. Nevertheless, in the last decade further reductions have been made to a figure of 1.25 lb per kw-hr. Also material progress has been made in developing equipment that can use low grades and various kinds of coal. The development of spreader-type stokers, water-cooled and slag-tap furnaces, cyclone combustion furnaces, and improved air preheaters and better understanding of the conditioning of boiler water to prevent scale formation and corrosion have provided very significant improvements in the operation of boiler plants.

With respect to residential and small commercial heating units, much progress has been made in recent years in providing smokeless and convenient automatic heat from solid fuels with stoker-fired, bin-feed, and ash-removal furnaces. Research on such equipment sponsored by the coal industry in the last 2 decades has borne fruit in recent years. Equipment including these improvements is now on the market, as are smokeless, hand-fired, magazine space heaters. These have become available at a time when a growing public demand for cleaner city atmospheres has led to prohibition of high-volatile bituminous-coal fuels unless burned in stoker-fired or other smokeless furnaces.

From the standpoint of future economic significance to the coal industry, a promising result of research in the field of combustion appears to be the powdered-coal combustion gas turbine sponsored by the Locomotive Development Committee of Bituminous Coal Research, Inc.[3]

A full-size coal-dust turbine locomotive is now well on its way to completion for test under railroad operating conditions. A byproduct of this research and development on coal utilization appears feasible in application of the coal-dust turbine to stationary use, particularly in areas of the country where water is not plentifully available. The coal-dust turbine requires little or no water in contrast with the conventional steam power-generating plant.

Carbonization and Gasification

Next to direct combustion of coal as a fuel, the production of coke, gas, and chemical products is the next largest field of utilization. In 1949, 80 pct of bituminous-coal production was burned for fuel purposes and 20 pct was used for carbonization and gasification.

At the Fourth World Power Conference in London in July 1950, it was evident from the papers and discussions that most nations that manufactured coke were concerned as to their reserves of coking coal. Many of them were conducting extensive research on making available their reserves of poorly coking coal by blending them with coals of better coking quality or by adding appropriate percentages of finely ground low-temperature coke, high-temperature coke breeze, anthracite fines, or pitch to the high-volatile coal charged into the ovens.

The United States also is facing the problem of extending its reserves of good coking coal by blending them with its much larger reserves of poorly coking coal to make a coke of adequate and usable quality; and of course this must be done before all of our high-quality coking coals are consumed. In most instances a better coke can be made by judicious blending of two or three different coals than from any one alone. This need is being recognized, and much attention is being given to research with experimental small and full-scale ovens, primarily for determining the physical properties of coke obtainable with various blends of coals and with different temperatures and rates of heating. This includes research on the expanding properties of the coal charge and the effect of various size consists of the charge. Such experiments by the Bureau of Mines and by industry were of immediate value when it became necessary to select coals from the Oklahoma-Arkansas field for supplying the wartime blast furnaces in Texas. Similar research on western coals was conducted for supplying the new blast furnaces in Utah and California.

During the war the general practice of blending low-volatile coal with high-volatile coal created a demand that was difficult to supply from available sources. This situation led to the use of anthracite fines at a number of plants to replace in part the usual proportion of low-volatile coal.[4] Such replacement gave satisfactory results at some of the plants as far as the physical properties of the coke were concerned, although it reduced the yield of byproducts. Since World War II some coke operators, especially those who were producing foundry coke, have continued using 2 to 8 pct anthracite fines to advantage.[5] Wartime research on blending coals was greatly facilitated by laboratory blending studies started before the war by several agencies.

The Bureau of Mines installed experimental slot-type ovens of about 400-lb capacity at its experiment stations at Pittsburgh, Pa., Denver, Colo., and Tuscaloosa, Ala. The Illinois Geological Survey likewise installed such an experimental oven and conducted extensive research in cooperation with industry on blending the higher oxygen Illinois coals with eastern bituminous coals and showed how this could be done with the production of suitable blast-furnace coke. The use of such experimental ovens and the Russell movable-wall oven at a number of commercial coke-oven plants has produced significant results in the improvement of coke quality by exploring the best possible blends of coals available. These investigations should lead to a considerable extension of our coking-coal reserves by including the lower ranks of poorly coking coal.

Closely related in importance to the washing and blending studies of coking coals in the last few years has been the Coke Evaluation Project sponsored jointly by the Iron and Steel Institute and the American Coke and Coal Chemicals Institute. Tests were conducted at various coke-oven plants with the aid of a portable testing laboratory installed on a truck. In this manner the same methods and equipment were used at the various plants, thus permitting comparisons heretofore impossible because of variations in sampling and testing. At the same time complete data on the analysis and size of the coal charged, the heat treatment of the charge, and the kind of ovens were obtained. While correlation of this work has not yet been completed, it is believed that significant practical results will be obtained.[6]

Research and development work on the Wisner process of low-temperature carbonization for the production of smokeless domestic fuel has resulted recently in the installation of a 1000-ton (coal)-a-day plant by the Disco Co. near Pittsburgh, Pa.[7] Progress also has been made in the use of low-temperature coke or char for improving the physical properties of metallurgical coke by blending some 15 to 20 pct of this low-volatile material with high-volatile coal at the plant of the Colorado Fuel & Iron Co. at Pueblo, Colo.[8] and at the Kaiser Co. plant at Fontana, Calif.[9]

During World War II increased demands for steel renewed interest in the possibilities of direct reduction of Minnesota iron ore with char or gas from North Dakota lignite. The Bureau of Mines began pilot-plant experiments on the carbonization and complete gasification of lignite and subbituminous coal.[10] It was found practicable to make a high-hydrogen gas that was especially suitable for the direct reduction of iron ore. A commercial-scale alloy-steel retort then was developed for further experimental operation to determine the best alloys for resisting corrosion and oxidation.[11] This development offers a means of using low-rank, noncaking coals for the production of high-hydrogen gas for synthetic ammonia, methanol, and other chemicals. Catalytic conversion of a part of this gas to methane would serve to produce high-Btu gas for pipeline transmission.

Another new development in low-temperature carbonization and to a greater extent in the complete gasification of coal has been the application of the "fluidized solids" technique[12] first employed in the catalytic cracking of petroleum to carbonization[13] and complete gasification of coal or lignite. In this process the bed of more or less finely divided coal becomes mobile and boils something like a liquid when a gas or vapor is passed through it. Such fluidization greatly promotes active contact between gases and solids, thus increasing rates of reaction and improving the temperature control of the process. This fluidization process works very well with lignite, subbituminous and noncaking coal but offers some difficulties with caking coal, which tends to soften and stick together. Thus far the best results have been obtained with noncaking materials; it is very promising in the production of carbon monoxide and hydrogen for synthetic liquid fuels, methanol, or synthetic ammonia.

Much research also is in progress on complete gasification processes where powdered coal is in suspension in a gaseous medium or in a fluidized bed. Oxygen and steam may be used as gasification mediums, and gasification at higher than atmospheric pressures promises greater economy than atmospheric pressure.

In England, the Gas Research Board has been conducting pilot-plant studies on the complete gasification of coal in fixed beds, using oxygen and steam under pressure and further raising the calorific value of the gas by catalytic conversion of part of the carbon monoxide and hydrogen to methane. The Board also is working on fluidized beds and on a process of hydrogenating the coal to produce a high-Btu methane-containing gas.[14]

In this country attention is being given to the production of a high-Btu gas from coal that would be suitable for supplementing natural gas when the supply becomes short or to furnish gas for peak loads. It is believed that such developments will become possible on the basis of work now in progress on the production of synthesis gas for liquid fuels.

The underground gasification experiments being conducted in the United States and abroad are necessarily of a long-range character. The results of the first and second experimental projects conducted by the Bureau of Mines in cooperation with the Alabama Power Co. show complete combustion of the coal bed as the fire progresses, and no difficulty has been experienced in maintaining it. However, much of the gas made is burned by short-circuiting of the air after the burned cavity becomes enlarged. This difficulty must be overcome to obtain a good producer gas. It might be possible to utilize the sensible heat, as well as the potential heat, in the gases thus generated by running the hot product gases into a gas turbine; as a matter of fact, this has been done at the Alabama experiment.

Synthetic Liquid Fuels

Finally in this review of recent coal research we come to the extensive program of research and development of the Bureau of Mines[15] and of industry in the production of synthetic liquid fuels. This work is bound to have significant economic results, not only in the provision of means for supplementing our supply of liquid fuels as our petroleum production declines below the constantly growing demand, but also for supplying chemical products from coal, and in opening up new fields of coal research that will have unforeseen commercial applications. Processes for the liquefaction of coal and for making synthetic liquid fuels were developed in Germany some 20 years ago. Both the Bergius-I.G. Farben. hydrogenation and the Fischer-Tropsch gas-synthesis processes were operated commercially in Germany before and during the war. These processes are now being improved and adapted to American conditions by Government and by industry. A 7000-bbl-a-day commercial plant for the production of gasoline, diesel oil, and chemicals from natural gas is beginning operation at Brownsville, Texas.

The Bureau of Mines is engaged in a comprehensive research and development program on oil from coal at Bruceton, near Pittsburgh, Pa., and on the production of synthesis gas from coal at Morgantown, W. Va. The gasification of lignite and subbituminous coal is being studied at Grand Forks, N. Dak., and at Denver, Colo. Research and development on oil from shale are being carried on at Laramie, Wyo., and large-scale development work by the Bureau is well-advanced on mining methods and retorting and refining processes at the oil shale deposit at Rifle, Colo.

Large-scale pilot-plant work on the production of oil from coal is proceeding at the Bureau's demonstration plant at Louisiana, Mo. A 200-bbl-a-day hydrogenation unit is in experimental operation, and a 50-bbl-a-day synthesis-gas or modified Fischer-Tropsch unit is nearing completion. Both of these pilot plants include new features and improvements developed by the Bureau's research groups and also are proceeding on the basis of research information obtained from German plants by American investigators following the close of hostilities. Test runs will be made on various ranks and types of coal from the different coal fields of the United States. The oils obtained will be in adequate quantity for working out appropriate methods of refining and recovery of valuable byproducts.

The Bureau of Mines demonstration mine at Rifle, Colo. has developed a highly mechanized method of mining a 70-ft-thick bed of shale that averages 30 gal of crude oil per ton. Operating costs in a recent test were about 30¢ a ton and overall costs are estimated at about 40¢ a ton. This low mining cost and a probably lower overall investment cost for mine and retorting plant than for coal-liquefaction plants favor oil-shale plants for being among the first synthetic liquid-fuel plants (aside from natural-gas plants) to compete with petroleum for supplying heating oils. At higher cost, shale oil may be converted to motor fuel by hydrogenation under pressure.

Interest in synthetic fuels production is growing. A number of oil and coal-processing companies are conducting research and development work on making oils from natural gas, coal, or oil shale. At present, cheap natural gas and oil shale seem to be the raw materials that will first approach the cost of natural petroleum products. However, further developments and rising prices of natural gas may bring coal and lignite into the picture earlier than is generally expected.

In addition to producing high-octane gasoline of good rich-mixture performance, the hydrogenation process can be made to yield a considerable proportion of benzene, toluene, xylene, phenol, and tar acids. These are now in short supply and are significant in view of the need of the synthetic rubber, plastics and chemical industries based on these materials. Defense requirements for these chemicals may lead to early construction of one or two coal-hydrogenation plants.

The Fischer-Tropsch or gas-synthesis process also may be regarded as an important new method of making useful chemicals from coal. In this case the chemical compounds belong to the aliphatic or methane series rather than the aromatic and napthenic series that characterize the products of the hydrogenation process. The Fischer-Tropsch byproducts are alcohols, aldehydes and organic acids. Both processes will have significant economic impacts on the organic chemical industry and, of course, on the coal industry when the need arises to supplement materially our supply of petroleum.

Summary and Conclusions

In concluding this review of the economic significance of recent technologic research on solid fuels in the United States, it appears that the achievements of economic significance are greater than is popularly supposed. Although the production of synthetic liquid fuel and high-heating-value gas from coal has not yet arrived commercially, definite progress has been made in this direction. In addition to these potential developments in preparing for future needs, research in mining, preparation and utilization of coal has led to improved mechanical production and more efficient utilization.

The mechanical cutting, mobile loading and conveying of coal in the mine are continually being improved, and the continuous mechanical mining of coal without the use of explosives is in successful commercial operation. The preparation of coal has been improved by research on methods so that finer sizes of coal can be treated, with resulting greater reduction of impurities and improved recovery of coal. The efficiency and convenient use of solid fuels have been increased in residential stoves as well as in large industrial power plants. The powdered-coal combustion gas turbine gives promise for locomotive power and also for electric power in arid regions where condensing water is inadequate for steam plants. Recent research on coking blends of good and poorly coking coals are increasing our total reserves of coal for the production of metallurgical coke, and two pioneering, commercial, low-temperature carbonization plants are extracting chemical products from coal and lignite and are providing smokeless solid fuels for domestic use. Finally, it is believed that the extensive research now in progress on the production of synthesis gas from solid fuels will lead to significant future improvements in the utilization of coal in directions not now appreciated.

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Concentration of the Complex Copper-Lead-Cobalt-Nickel Ores of Southeast Missouri

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THE results of a research and development laboratory and pilot-plant mineral-dressing investigation by the Bureau of Mines of the complex copper-lead-cobalt-nickel ores of southeast Missouri are reported in this paper. This investigation, in cooperation with the National Lead Co., showed that separate concentrates of lead, copper, and cobalt-nickel could be made by fine grinding and flotation. An industrial plant built subsequently by the St. Louis Smelting and Refining Div. near Fredericktown, Madison County, Mo., produced such concentrates during the war years. Present commercial production at the mill is limited to lead and copper concentrates, but research to permit economic utilization of the cobalt and nickel values is continuing.

The complex sulphide deposits near Fredericktown in Madison County, Mo., have an operating history that dates back to the years preceding the Civil War. Perhaps these operations can best be summarized in terms of output which are given by Tyler,[1] "Previous to 1900, the Mine la Motte Co. shipped cobalt ore to Swansea and in 1903 produced 120,000 pounds of oxide in a Missouri plant. Another cobalt refinery erected in 1906 by the North America Lead Co. at Fredericktown, Mo., operated intermittently until 1910. In 1909 it produced 83,384 pounds of cobalt oxide, 328,403 pounds of nickel, 8,314 tons of nickel-cobalt concentrates, 600 tons of copper concentrates, and 1,353 tons of lead concentrates. This plant was purchased at a forced sale in 1915 by the Missouri Cobalt Co., which produced a certain amount of oxide in 1918 and in 1920 recovered 102,410 pounds of cobalt oxide and cobalt hydrate, valued at $262,810."

According to Buehler,[2] "The ore, which consists of a complex mixture of the sulphides of cobalt, nickel, copper, lead, and iron, occurs at the contact of the pre-Cambrian porphyry and the overlying Lamotte sandstone and Bonneterre dolomite." The source of most of the cobalt and nickel is siegenite, a member of the linnaetite series of minerals. Its composition is variable, and although the chemical formula may be given as $(CoNi)_3S_4$, it usually contains both iron and copper. The siegenite is characterized in this instance by an intimate association with chalcopyrite, and to a lesser extent with iron sulphides. Galena, on the other hand, while disseminated throughout the ore, is not closely bound to other constituents. Gangue is composed essentially of dolomite and quartz.

Petrographic examination showed that fair liberation of sulphides and gangue could be attained at —100 mesh. Complete liberation of micronsize inclusions, particularly of sulphides in each other would not have been possible at —400 mesh.

Beneficiation Studies

Many samples of the southeast Missouri ore were tested, but most of the work was confined to five samples of churn drill cuttings weighing 15 tons. The samples were reportedly representative of five sections of the deposit and were tested individually and as a composite. Only the results of the tests on the composite, which contained 1.28 pct lead, 2.03 pct copper, 1.06 pct nickel, 0.87 pct cobalt, 3.57 pct iron, 15.4 pct lime, and 23.9 pct silica, are presented in this report.

In view of the rather complex and intimate association in the mineralogical composition of the ore, flotation was the logical choice in the selection of a beneficiation process. Other mineral dressing unit operations were considered as an adjunct to flotation, but none were beneficial. Many tests were made on these samples to determine the flotation characteristics of the mineral siegenite. As a result of this work, it was noted the siegenite was intermediate in floatability between chalcopyrite and galena on one extreme and iron sulphides on the other. The difference in floatability between siegenite and the iron sulphides was not very great and, in general, the same reagents that depressed pyrite and marcasite would also, in slightly larger quantities, depress siegenite. This tendency for siegenite

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and pyrite to overlap made the concentration of the ores high in pyrite extremely difficult. The determining factor was actually the ratio of pyrite to siegenite in the sample. That is, with a low to medium ratio, fairly good recoveries were possible by flotation at good grades; in other cases, the reverse was true.

The most practical reagent suite included lime and cyanide as depressants for the rest of the sulphides while floating galena and chalcopyrite with an Aerofloat or lower xanthate. The siegenite was collected later by means of carefully regulated additions of a stronger promoter such as a higher xanthate. Judicious application of the promoter at that point was necessary to prevent undue flotation of the pyrite. Single cleaning of each concentrate with no additional reagents was the usual practice.

The results of a typical batch flotation test at a grind of —100 mesh on the composite sample are presented in Table I. The recovery of copper was excellent and that of the lead was fairly good in a concentrate containing 19.5 pct copper and 12.6 pct lead. Fair recoveries of both cobalt and nickel were made in another concentrate containing 14.5 pct cobalt and 16.3 pct nickel.

Pilot-Plant Operation

To confirm the results of the batch testing and provide the cooperating company with a large sample of the cobalt-nickel and copper-lead concentrates for further experimentation, pilot-plant flotation tests were made. The pilot-plant was operated 24 hr a day, except for minor interruptions for repairs and disposal of tailing.

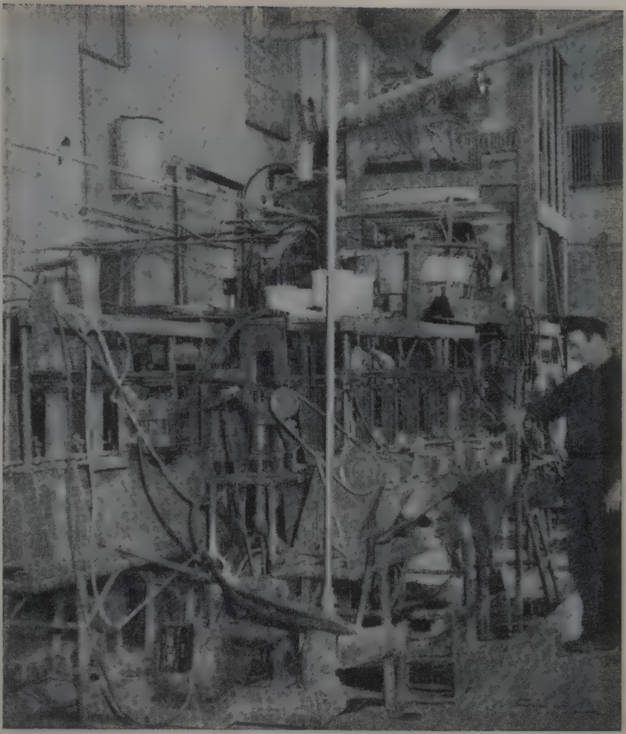

Fig. 1—Flotation pilot plant.

In the pilot plant, the crushed ore was ground at an average rate of about 3.0 lb a min in a 19x36-in. ball mill in closed circuit with a 24-in. bowl classifier. As in the batch tests, lime and cyanide were added to the ball mill. The classifier overflow was laundered to a 12-in. conditioner, to which either Aerofloat 15 or 31 was added as a promoter for the copper and lead minerals. The pulp flowed by gravity to a bank of six Denver No. 5 flotation cells, where the chalcopyrite and galena were removed in bulk. The pulp leaving the copper-lead circuit was pumped to a similar bank of cells, where xanthate Z-8 and a frother were added and the siegenite was roughed off in the first three cells and cleaned once in the others. The final tailing was rejected in the cobalt-nickel circuit. The flotation pilot plant is shown in Fig. 1.

Table I. Results of Batch Flotation Test

Metallurgical Data

Product	Weight, Pct	Analysis, Pct Pb	Cu	Ni	Co	Pct of Total Pb	Cu	Ni	Co
Copper-lead concentrate	7.7	12.6	19.5	1.8	1.4	74.1	92.6	14.0	12.8
Cobalt-nickel concentrate	4.1	2.0	0.30	16.3	14.5	6.3	0.8	67.7	70.8
Copper-lead cleaner tailing	1.5	3.0	1.0	1.2	0.70	3.4	0.9	1.8	1.2
Cobalt-nickel cleaner tailing	1.4	3.0	0.50	3.1	3.0	3.2	0.4	4.4	5.0
Rougher tailing	85.3	0.20	0.10	0.14	0.10	13.0	5.3	12.1	10.2
Composite	100.0	1.3	1.6	0.99	0.84	100.0	100.0	100.0	100.0

Operating Data

	Lb per Ton of Ore						
	Copper-lead Circuit				Cobalt-nickel Circuit		
Reagent	Ball Mill	Conditioner	Rougher	Cleaner	Conditioner	Rougher	Cleaner
Lime (burned)	3.0						
Sodium cyanide	0.20						
Aerofloat 15		0.12					
Xanthate Z-8					0.075		
Cresylic acid					0.12		
Time, min	20	5	5	4	5	5	4

Results typical of a single shift during the pilot run are summarized in Table II. The recoveries of lead and copper were 87.1 pct and 81.6 pct in a concentrate analyzing 20.6 pct copper, 10.1 pct lead, 0.99 pct nickel, and 0.77 pct cobalt. This represented a material increase in the recovery of lead over that attained in the batch tests, but the recovery of copper was lower. At the time the pilot plant was in

Table II. Typical Results of Pilot-plant Testing

Metallurgical Data

Product	Weight, Pct	Analysis, Pct Pb	Cu	Ni	Co	Pct of Total Pb	Cu	Ni	Co
Copper-lead concentrates	7.9	10.1	20.6	0.99	0.77	87.1	81.6	7.7	8.8
Cobalt-nickel concentrates	5.1	1.3	5.5	14.4	9.3	7.2	14.1	71.9	68.6
Tailing	87.0	0.06	0.10	0.24	0.18	5.7	4.3	20.4	22.6
Composite	100.0	0.91	2.0	1.0	0.69	100.0	100.0	100.0	100.0

Reagents Consumption, Lb per Ton:

Lime (hydrated)	1.2
Sodium cyanide	0.23
Xanthate Z-8	0.14
Cresylic acid	0.19
Aerofloat 31	0.19

Table III. Results of Plant Operation

Product	Weight, Pct	Analysis, Pct					Pct of Total				
		Pb	Cu	Ni	Co	Fe	Pb	Cu	Ni	Co	Fe
Lead concentrate	2.88	75.97	2.66	0.21	0.16	3.26	90.61	6.95	1.53	1.51	1.15
Copper concentrate	3.28	2.81	22.76	2.45	1.86	26.72	3.81	67.68	19.91	19.79	10.85
Cobalt-nickel iron concentrate	4.56	1.50	2.77	4.28	3.26	35.74	2.82	11.46	48.35	48.10	20.15
Tailing	89.28	0.07	0.17	0.13	0.10	6.14	2.76	13.91	30.21	30.60	67.85
Mill feed	100.00	2.41	1.10	0.40	0.31	8.08	100.00	100.00	100.00	100.00	100.00

operation it was noted that chalcopyrite was not floating as readily as in the small tests. The latter had indicated that 8 min flotation time would be required for the copper-lead flotation. The six rougher cells in that circuit provided 8 min of contact time, and it was adequate for the galena, but rapid flotation of the chalcopyrite equal to that of the batch tests was not achieved. Flotation capacity was limited, and it was not possible to increase the number of cells to correct this situation. Other possible corrective measures, such as changes in the reagent combination, were investigated, but none were successful.

Considering other results of the pilot test, total recovery of nickel plus cobalt was greater than in batch tests, but the grade of concentrate was lower. The latter was due in part to the fact that chalcopyrite, which had not floated in the copper-lead circuit, reported in the cobalt-nickel concentrate.

Why chalcopyrite was more reluctant to float in the pilot plant than in the batch tests is difficult to explain. Differences in grind and soluble salt content may have contributed to the difference in behavior. In retrospect, it is probable the pilot results could have been improved by reducing the feed rate to provide additional contact time in the flotation circuit. While this is not an entirely satisfactory solution, such a move might have improved the copper recovery and both grade and recovery of the cobalt-nickel product.

Limitations in the amount of available flotation capacity also prevented inclusion of a copper-lead separation circuit in the pilot plant. However, batch tests made later on a sample of pilot-plant copper-lead concentrates showed that a separation could be effected either by 1—depressing galena with an alkali dichromate and floating the chalcopyrite, or 2—depressing the chalcopyrite with a cyanide and floating galena.

National Lead Co. Operations

Early in 1944, the National Lead Co. erected and placed in operation a 600-ton flotation mill near Fredericktown, Mo., the products of which were concentrates of lead, copper, and cobalt-nickel. The latter product, which was roasted prior to shipment, was reported to contain 19 pct nickel and 13 pct cobalt.[3] These products comprised the output of the mill until the end of World War II at which time production of high grade cobalt-nickel concentrates was suspended as a result of reduced demand for the two metals.

The Madison County mill has continued to operate but with emphasis on the production of metals other than cobalt and nickel. The analysis of recently milled ore, for instance, shows 2.41 pct lead, 1.10 pct copper, 0.40 pct nickel, 0.31 pct cobalt, and 8.08 pct iron.

Present practice in the mill calls for crushing and grinding to —65 mesh. A bulk flotation of all sulphides (other than part of the pyrite and marcasite) is effected at the outset, after which the application of a cyanide permits continued flotation of the galena only. The galena is cleaned and removed from the circuit. Remaining sulphides are thickened, reground to about —325 mesh and refloated. The latter operation effects a separation into a copper concentrate, which is sold, and a cobalt-nickel-iron reject, which is impounded on the premises. Results typical of 1950 operations are presented in Table III. Since those data were gathered, improved operations have resulted in a copper grade of 28.0 pct at substantially the same recovery noted in Table III.

A direct comparison between the plant data and the results of the Bureau of Mines pilot plant is not possible because of differences in procedure and analyses of the respective feeds. Certain similarities, however, are evident in both sets of data. Among these is the fact that galena is the most floatable constituent, so that an excellent recovery of lead is attained. A lesser recovery of copper is achieved in the copper concentrate, and much of the chalcopyrite that fails to report in that product is recovered later in the cobalt-nickel-iron concentrate.

Acknowledgment

Acknowledgment is made to E. S. Allen, former Mill Superintendent, St. Louis Smelting and Refining Div., National Lead Co., for supplying operating data of the Fredericktown, Mo., plant, and to Gloyd M. Wiles, Manager, Mining Dept., National Lead Co., for permission to publish those data.

Summary and Conclusions

The results of laboratory and pilot-plant investigations on the beneficiation of the complex copper-lead-cobalt-nickel ores showed that separate concentrates of lead, copper, and cobalt-nickel could be produced by fine grinding and flotation, a conclusion confirmed by the subsequent operation of the Fredericktown, Mo., plant of the National Lead Co.

Present output of the plant is limited to lead and copper concentrates as there has been no commercial production of cobalt and nickel since the end of World War II. The recent increase in demand for both metals for defense needs has instigated a considerable amount of research by government and private agencies into the utilization of southeast Missouri concentrates as a source of cobalt and nickel. The successful completion of that research probably will result in the resumption of commercial cobalt-nickel production in Missouri.

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Shallow Expressions of Silver Belt Ore Shoots Coeur d'Alene District, Idaho

by Robert E. Sorenson

EXPLORATION for deep-seated orebodies in the Silver Belt area of the Coeur d'Alene mining district is complicated by meager surface expressions of diagnostic criteria, lack of knowledge of the significance of specific alteration phenomena, and the great expense involved in the exploration for ore shoots, the tops of which often occur near sea level elevation or below and as much as 4000 ft or more below the surface. Except for the early studies of Ransome and Calkins in 1903 and 1904,[1] little geological work was done in the Coeur d'Alene district for many years, and it is only during the last 15 to 20 years that complete and careful geological mapping has been applied to most of the mines in the area. Much early information, which now would prove invaluable, is lost in hundreds of miles of caved and inaccessible mine openings not mapped by mining geologists. In more recent years mining geology has become a tool of greater use in the district, and some measurable progress is being made in the accumulation of observations that are adding gradually to the knowledge of ore deposits in the district.

The recognition of alteration phenomena and related features by Hoyt S. Gale[3] in 1935, followed by the work of Shenon and McConnel in 1936-37,[4] changed many of the concepts formerly held and opened the way for more critical study and mapping of alteration effects.

Mapping of these effects has been in progress for several years but no well-organized, careful, and complete laboratory study of the alteration has been made by mining companies or by government agencies. Recently, Thomas Mitcham of Columbia University has been occupied in the collection and study of specimens carefully selected from several producing mines in the Silver Belt as well as from several prospects exhibiting alteration effects. Mr. Mitcham and the writer have discussed some of the preliminary results of his investigations, which indicate that definite and interesting material is forthcoming from his studies. Work of this kind, aided by careful observation by district geologists, should advance the knowledge of the environment of Silver Belt ore shoots and establish additional criteria for recognizing meager shallow expressions of deep-lying orebodies.

Structural details in the Silver Belt area are extremely difficult to decipher on the surface because the area is almost completely covered by a deep mantle of surface debris and soil. A dense growth of small trees and buck brush adds to a geologist's difficulties. Often large areas reveal only scattered small outcrops with great intervening areas where the underlying formations must be deduced from small altered rock fragments. Some considerable success in obtaining surface information has come from bulldozer roads and cuts in selected areas. As this type of work receives more attention, knowledge of the surface expressions of formations, alterations, faulting, and veining is expanding and giving more reliable information to guide deeper development. Lack of diagnostic marker beds in the district contributes to a geologist's difficulties in determining fault displacements.

In the Silver Belt, known commercally important sulphide orebodies commonly are associated with siderite and quartz gangue. Some of the veins at shallow horizons above productive ore shoots are composed of quartz with light-colored carbonate minerals such as calcite, ferrocalcite, ferrodolomite, ankerite, and carbonates with variable amounts of manganese. The relation of these carbonates to commercial orebodies is a problem offering possibilities for research. Shallow exposures of veins composed of quartz and the light-colored carbonates, often with some associated barite, grade into quartz and siderite veins at deeper horizons. The vertical range of these carbonates has been partially established for some veins, but no reliable method has been devised for predicting where the change in a vein may take place. Careful observation and analysis of carbonates may make it possible to predict more closely where the change to siderite may be expected. In 1926 Shannon[5] pointed out some of the characteristics of the carbonate veins of the district and included a few chemical analyses.

Some of the Silver Belt ore shoots have small commercial outcrops, and very shallow development of some properties has led to the opening of small ore shoots.[6] Several small ore shoots developed at shallow depths have not been explored at deep horizons, and some important ore shoots with no commercially significant exposures at shallow horizons have been developed. The greatest problem of the mining geologist working in the Silver Belt area is to find the all-important link between the very meager shallow expressions of mineralization and the rich ore shoots that may be found below. To develop criteria for use in the recognition of the weak upper expression of ore shoots, the experience of past development programs must be relied upon heavily. The investment of more than a million dollars in the exploration for such ore shoots requires some reliable criteria if these ore shoots are to be found. Management needs some assurance that the risk involved is justified by well-developed and acceptable geological inference. The richness of some of the ore shoots found thus far has contributed greatly to the justification for a bold approach, but many negative results of blind exploration programs would lead eventually to extreme caution on the part of management. Fortunately, recent experiences of development programs have been quite successful. It is hoped that added knowledge of the environment of Silver Belt ore shoots will contribute greatly to the sureness of approach toward their development.

The Coeur d'Alene mining district is located in the panhandle of Idaho in Shoshone County. The population of the county is either employed in the

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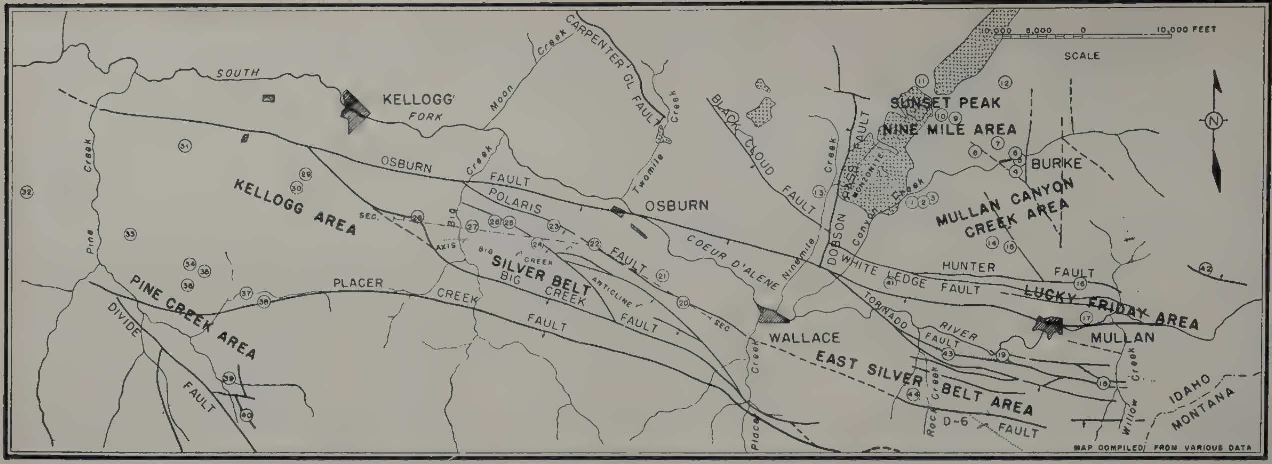


Fig. 1—Coeur d'Alene mining district showing mines and major faults.

1—Gem
2—Frisco
3—Black Bear
4—Hecla
5—Poorman
6—Tiger
7—Sherman
8—Standard Mammoth
9—Custer
10—Tamarack
11—Interstate
12—Hercules
13—Day Rock
14—Star
15—Morning
16—Gold Hunter
17—Lucky Friday
18—Atlas
19—Moe
20—Galena
21—Argentine
22—Coeur d'Alene Mines
23—Nellie
24—Silver Summit
25—Chester
26—Polaris
27—Sunshine
28 Crescent
29—Last Chance
30—Bunker Hill
31—Page
32—Hypotheek
33—Liberal King
34—Denver
35—Sidney
36—Pittsburg
37—Nevada Stewart
38—Highland Surprise
39—Douglas
40—Constitution
41—Golconda
42—Snowstorm
43—Gem State
44—Rock Creek

mining of lead, zinc, and silver or serving the requirements of those occupied in mining. The history of the district dates back to the first discovery of gold in the Murray district in 1879. The gold miners and prospectors from the Murray area roamed the Coeur d'Alene district in the search of more gold and in the early 80's discovered some of the lead-silver-zinc veins of the Mullan, Burke, and Kellogg areas. Minor production of the base metals began in the 80's, and annual production reached ten million dollars in 1900.[2] The district has continued to be a large producer of lead, zinc, and silver. Several years ago the district passed the billion dollar mark in the value of its output.

The distribution of the mines is shown on Fig. 1, which is compiled from several sources.[6] Much of the early production came from the Canyon Creek, Mullan, and Nine Mile areas northeast of Wallace and northwest of Mullan, and from the Kellogg area which contains the great Bunker Hill and Sullivan mine. The Pine Creek area located southwesterly from Kellogg gained more importance as the interest increased in zinc, the most important metal in Pine Creek.

Although the Silver Belt, shown in Fig. 2, contains some of the early lode locations of the Coeur d'Alene district, early production was restricted to very small near surface shoots which produced little or no profit to the early day miners. The Sunshine mine was worked in a small way until the late twenties, and persistent greater depth development of the ore shoots found near the surface, resulted in the development of one of the great silver mines of the world. Sunshine has paid more than 29 million dollars in dividends.

Success of the Sunshine venture brought about exploration of the Polaris mine adjacent to the Sunshine. The Coeur d'Alene Mines Corp. explored and mined a small ore shoot; Polaris Mining Co. through easterly workings of the Sunshine mine found the Rambo-Omega-Rotbart orebody containing more than 60 million oz of silver and 60 million lb of lead; and in 1948 Silver Summit Mining Co., assisted by financial help of Polaris Mining Co., found a deep-seated ore shoot on the 3000 level approximately 340 ft below sea level.[7] Since that time the vein was developed on 3200 and 3400 levels, and mining is proceeding at a rate of about 4000 tons per month. In the latter part of 1950, in a new deep shaft on the property of Vulcan Mining Co. on Lake Gulch just west of Wallace, the American Smelting and Refining Co. and Day Mines, Inc. began opening an extremely interesting orebody approximately at sea level elevation. The expansion of the Silver Belt proceeds easterly at an accelerated rate with new discoveries.

Geology

The principal rocks of the district are argillite, quartzite, and intermediate types comprising the Belt series of pre-Cambrian metamorphosed sediments widely distributed in northern Idaho, western Montana, eastern Washington, and southern British Columbia. In the Coeur d'Alene district this series consists of more than 20,000 ft of sediments of shallow-water origin that have been divided into:

1—The Prichard Formation, consisting of more than 12,000 ft of black argillite and argillaceous quartzite.[4]

2—The Burke Formation, approximately 1800 to 2400 ft thick, composed of gray to white quartzite and argillaceous quartzite.

3—The Revett Formation, 2100 to 3400 ft thick, comprising massive thick-bedded gray to white quartzite with some interbedded argillaceous quartzite.[4]

4—The St. Regis Formation, variously estimated as 1000 to more than 2000 ft thick and consisting of thin-bedded purplish-gray to purple argillite and quartzite.

5—The Wallace Formation, estimated by Shenon and McConnel to be 4500 to 6000 ft thick in the Silver Belt[4] composed of gray-black calcareous argillite and white to gray calcareous quartzite.

6—The Striped Peak Formation, consisting of 1500 ft or more of purplish, pink and green quartzite, sandy quartzite and calcareous argillite, the youngest rock of the Belt series and not a host rock for any of the orebodies discovered thus far.

These sediments have been variously metamorphosed by dynamic processes with the development of abundant sericite and consolidation of the original rocks. The St. Regis and Wallace formations are the most prevalent formations seen on the surface in the Silver Belt and east to the Montana-Idaho state line.

The most abundant intrusive rock is the monzonite and syenite shown as small stocks and irregular masses in Fig. 1. Some associated dikes form protruding tongues in the sediments adjoining the monzonite. Contact metamorphic alteration of the sediments around the periphery of intrusives is common.[1]

Premineral diabase dikes and postmineral lamprophyre dikes, both commonly associated closely with large faults and with areas of mineralization, are widely distributed in the district.

Folding has greatly affected the rocks of the district. North of the Osburn fault the axes of folds generally trend northerly to northwesterly, and south of the fault the folds trend more nearly east-west. In the Silver Belt area the most prominent fold is the Big Creek anticline, shown in Figs. 1 and 2. It is a complex fold with a gently dipping south flank, an overturned axis, and a partially overturned north flank. Several minor folds occur on its steep north limb. In the East Silver Belt, see Fig. 2, another similar strongly developed and complex anticlinal structure has been mapped. There appears to be an intimate relationship between the stronger mineralized areas and the axis and the north flank of the Big Creek anticline. The prominent fold in the East Silver Belt area has similar relationships to much of the veining and sulphide mineralization that is known in this sparsely developed section of the district.

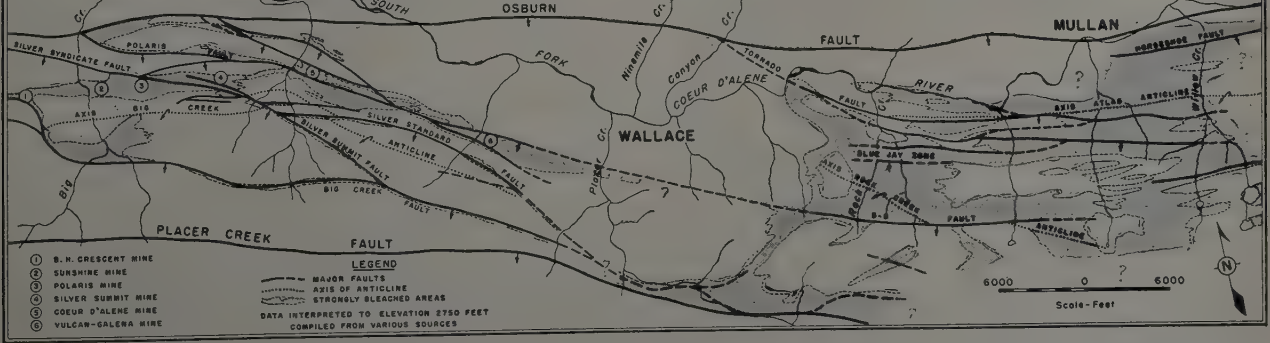
Faulting in the Coeur d'Alene district has been described by Ransome and Calkins,[1] Umpleby and Jones,[2] and Shenon and McConnel.[4] The great Osburn fault, which cuts an easterly-westerly course through the Coeur d'Alene district and which may be mapped for scores of miles east and west of the district, is known to most geologists as one of the world's great faults. It is a normal structure with a probable great horizontal component and has a southerly dip of 55° to 65°. Commonly, several hundred feet of crumpled rock, gouge, and breccia mark the structure, and it is expressed on the surface by a series of characteristic saddles on ridges crossed by the structure. On the north side of the Osburn fault other large fault structures have been described in the literature,[1,2] and a few of the major structures are shown in Fig. 1. In the area east of the monzonite intrusive and north of the Osburn fault, the predominating pattern is one of block faults.

South of the Osburn fault in the Silver Belt area some of the principal fault structures are shown in Figs. 1 and 2. One of these is the Placer Creek fault, a south-dipping normal fault located two to three miles south of the Osburn fault and trending roughly parallel to the latter across most of the district. Other important faults, all south-dipping, include the normal Polaris fault, the reverse Silver Summit or Silver Syndicate fault, the reverse Chester fault (vein), and the reverse Big Creek fault, all structures of persistence showing variable displacements generally measurable in hundreds to thousands of feet.[1,3] The above-mentioned structures, plus other linking faults, break the area into blocks of variable relative displacement. The tensional effects set up by differential movements on the major structures locally developed openings through which the mineralizing solutions were able to pass and in which many of the presently known veins were deposited by ascending solutions.

Although similar structural conditions have been mapped in the East Silver Belt by the U. S. Geological Survey[8] and by company geologists, the structure is not as completely known in this area because of meager underground development and poor exposures. Tentatively the D-6 fault has been correlated with the Polaris fault to the west and other strong faults, and associated vein-filled tensional fractures have been mapped in the eastern part of the area, see Fig. 2.

Alteration of the argillite and quartzite of the Silver Belt has been referred to in this paper and references were made to the original recognition of the "bleached" rocks by Gale[3] and by Shenon and McConnel.[4] This subject matter has been discussed by the author,[6,7] and Hobbs, Wallace, and Griggs mapped and described the bleaching in their preliminary report on the East Silver Belt area. As mentioned previously, careful investigation of the processes and chemistry involved has not been completed but such work is in progress. Also, Hobbs, Wallace, and Griggs[8] make a short reference to abundant chloritic alteration in the vicinity of the Atlas mine. This chloritization was mapped and described by the author in 1932 in connection with a study of the area for the Hecla Mining Co. Subsequent deeper exploration of the Atlas by the Hecla

Fig. 2—Silver Belt of the Coeur d'Alene mining district showing major faults and strongly bleached zones.



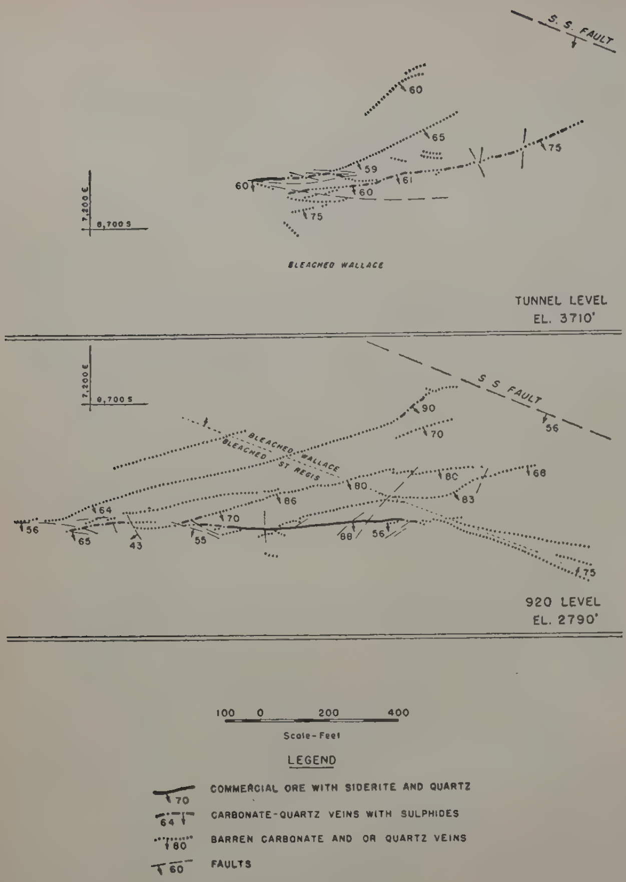


Fig. 3—Polaris mine.

Mining Co. revealed additional relationships between the chloritization and the more widespread bleaching. In 1941, in a private report to Hecla, P. J. Shenon pointed out the relationship of the chloritization and bleaching.

Chloritization in the vicinity of the Atlas mine is distributed in lens-like masses from a few feet to 500 ft wide and up to a mile long. Considerable chloritic quartzite is evident for about four miles westerly from the Atlas and more than a mile toward the east. The chloritized quartzite and argillite vary from pale green to intensely dark green depending on the amount of chlorite present. Whitish specks are disseminated through the rock in many areas giving the rock the appearance of a porphyritic texture. Weathering of the whitish particles gives them a lightly iron-stained coloration. In the thin section the rock is composed of rounded quartz grains showing secondary enlargement, sericite, carbonate, and chlorite. Chlorite and carbonate appear to be introduced substances, and with sericite they form the interstitial material between the quartz grains of the original sediment. The erratic distribution of the carbonate and chlorite, together with the fact that the chloritic zone crosscuts the quartzite on strike and dip, further sustains the viewpoint that the substances were introduced.

Deeper development of the Atlas area revealed that the chloritic rock, along with purplish sericitic quartzite of the St. Regis formation, was bleached by the later process, which developed the large strong bleached zones in the Silver Belt. The chloritic phase, then, was the early manifestation of hydrothermal alteration, probably a somewhat lower temperature phase as pointed out by Shenon in his Atlas report of 1941.

The bleaching process has been described in a general manner as it pertains to the Coeur d'Alene district but specific careful study remains to be done. Thus far no positive evidence has been developed to support the theory that an abundance of potash has been added to the original rocks, and the answer will come only from careful work. Sericite is generally present in large quantities both in unbleached and bleached rocks of the district. Megascopically, however, it appears that sericite is more abundant in the bleached rocks, and some of them might be classified properly as fine-grained sericitic schists.

Essentially the bleaching process resulted in the removal of the coloring matter in the original rock, thereby developing prevailing light colors varying from almost white through whitish-green to apple green. In some instances, especially in the finer-grained argillaceous rocks, original rock structures and textures are destroyed. Bedding may be unrecognizable or a pseudo-bedding may develop because of the prevailing direction of the bleaching that often follows shearing, fracture cleavage, or joints in the rock. Commonly, however, bleached bands follow bedding.

The purplish coloration of the St. Regis formation, largely caused by the presence of finely divided hematite, is entirely removed in thoroughly bleached areas, and it is the author's opinion that much of the iron of the abundant pyrite, disseminated in the bleached zones and often localized in certain areas within the bleached zones, had its source in the finely divided hematite dissolved from the rocks and carried upward by the bleaching medium.

The original coloring matter of the gray to black Wallace argillite is not defiinitely identified, but the bleaching process likewise removes all coloring matter from these rocks.

Although abundant disseminated carbonate, including ankerite, siderite, and possibly dolomite, are common in the bleached areas, sufficient work has not been done to clarify the relationship between the presence of the carbonates and the work of the bleaching medium. The carbonate may be more closely related to veining, which appears to succeed the bleaching in age.

Although the bleaching effects were noted first in the Silver Belt, and most carefully mapped there, the same alteration is prevalent in the Canyon Creek-Mullan area, the Kellogg area, and other sections of the Coeur d'Alene district and its extensions easterly and westerly. In the dark-colored Prichard argillite and quartzite bleaching appears to have effected less pronounced changes of the rock, and zones of bleaching in the Prichard formation are less prevalent than in the other formations.

Bleached zones tend to follow prominent structural trends, including crumpled flanks of folds, fault structures, shear zones, and other zones of disturbance. Locally, the more minor structures such as bedding, minor shearing, fracture cleavage, and jointing control the distribution of bleaching.

In the Silver Belt and its easterly extension the distribution of the zones and areas of bleaching are shown in Fig. 2. The most prominent and intense zone of bleaching follows a crescent-shaped course for several miles. The distribution shown on the map is necessarily somewhat diagrammatic because of imperfect exposures, but the prevailing trends of the zones have an easterly-westerly course through the area, and the widths are extremely variable. The sources of the bleached pattern and much of the structure shown on the map include Shenon and McConnel's map of the Silver Belt,[4] Hobbs, Wallace, and Griggs' map of the East Silver Belt,[8] and company records and maps available to the writer. Although attempts have been made to differentiate between various intensities of bleaching, it appears probable that the bleaching process affects certain rocks quite differently than it does others. Intensity may be mapped within certain limits, but it is not definitely known what weight may be given to the evidence. Currently, the attempt is to make a differentiation of this kind with the hope that such observations will serve some purpose as knowledge expands.

Veining in the Silver Belt area appears to be more prolific within the bleached areas, but several veins are found in unbleached rocks. Although some commercially important mineralization is associated intimately with some of the major faults, for example, the Silver Syndicate, the Chester, and the Alhambra faults, the most important orebodies found thus far favor tensional fractures that have resulted from stresses set up by the differential movement on the larger structures. Following the bleaching, these tensional type fractures were opened by structural adjustment, and carbonate-quartz-pyrite veins filled these fractures. Subsequent adjustment reopened the veins for the entrance of the sulphides. Quartz and siderite accompanied the sulphides, and a late stage of quartz succeeded their deposition. Sequence of sulphide deposition in most of the veins studied appears to have been pyrite, arsenopyrite, sphalerite, chalcopyrite, freibergite, and galena. The position of chalcopyrite in the sequence is not shown clearly in all of the instances studied. It often appears earlier than the silver-rich tetrahedrite, but a contemporaneous and even a later deposition is often indicated in the polished surface.

Subsequent to the sulphide stage of mineralization, a late phase of chloritization developed minor chlorite on small faults, minor shears, and joints. Occasionally the mineral has a dusty appearance in minor openings, and in several instances it appears to be a halo mineral around ore shoots. This relationship might be expected if the mineral is the dying ebb of mineralization.

It has been suspected by most geological workers in the Coeur d'Alene district that the monzonite intrusives present in the Canyon Creek-Nine Mile area are merely small protuberances of a large batholith underlying most of the district. The widespread areas of alteration in the Canyon Creek-Mullan area testify to the presence of such an underlying source of the bleaching medium, and on this north side of the Osburn fault, zones of strong bleaching may be followed to the Montana line and into that state for tens of miles. Likewise south of the Osburn fault in the Silver Belt, East Silver Belt, and on into Montana toward the east, similar zones of alteration persist with unabated strength. Diabase and lamprophyre dikes appear most prolific in these same areas, the former preceding and the latter succeeding the period of greatest sulphide development. The distribution of these features is more than a coincidence. Strong, persistent, depth-reaching fault structures like the Osburn, Polaris, Big Creek, and Placer Creek faults with reoccurring structural adjustment opened the area and maintained openings through which the mineralizing medium was able to pass toward the surface and produce the above results.

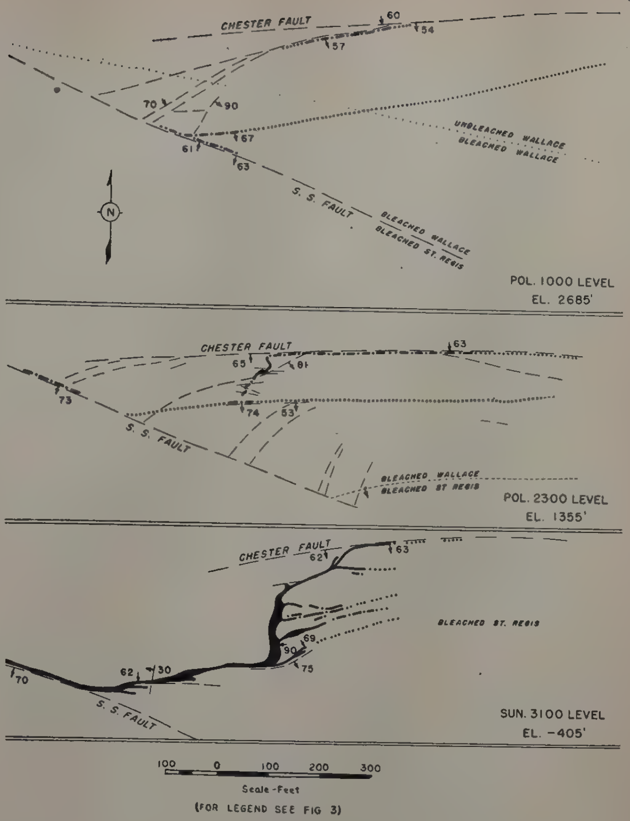


Fig. 4—Omega-Rotbart Veins, Sunshine and Polaris mines.

Although much evidence of alteration, faulting, and veining is obscured by an overabundance of debris, soil, dense brush, and timber, geologists are able to select zones of more prolific alteration on the surface. Evidence of veining often may be found, and inexpensive bulldozer work of recent years serves to clarify the situation and allows more careful structural mapping to be done.

Because exploration for Silver Belt ore shoots must reach to depths requiring a large amount of underground work and more than the usual amount of expense incurred in finding orebodies, geologists of the district have been groping for shallow evidences of ore shoots. The shallow expressions of some of the known orebodies have been so meager that only the most daring approach has been possible in the past, but deliberate and extremely careful observation, mapping, and research may serve to bridge the gap between a blind approach and a well-planned development program tied to the probable structural environment of the type of orebodies sought and closely related to factual material available to the careful observer.

Geologists must depend on the background of evidence resulting from previously successful exploration programs and take heed from negative criteria available in the study of unsuccessful ventures when seeking new and unknown orebodies. It is important that the approach to new developments include careful observation that may serve to enlarge the scope of the science and develop more complete evidence for direct application to the search for deep-lying, hidden ore shoots.

In the Silver Belt the structural environment differs in the several productive properties, but the general statement may be made that the ore shoots

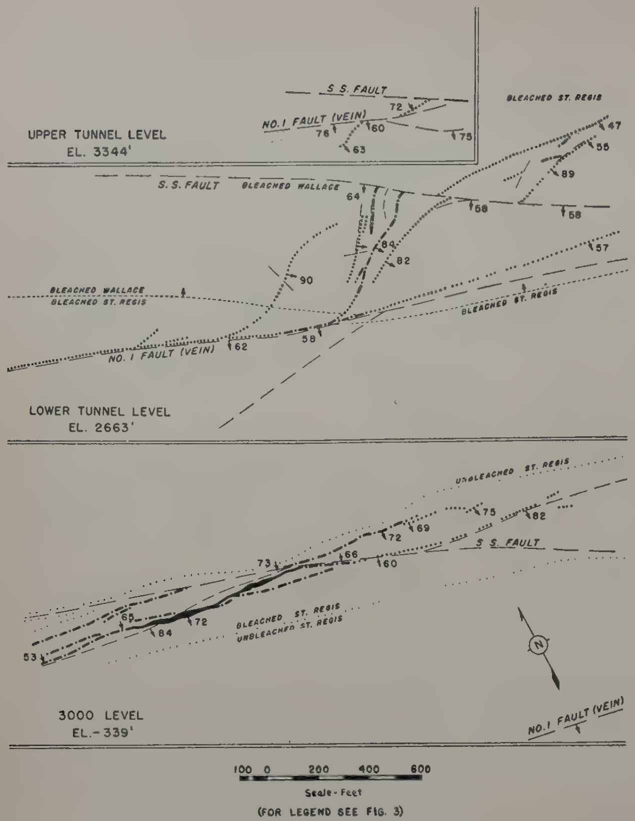


Fig. 5—Silver Summit mine.

have been found at depth below shallow expressions of strong structural conditions with associated zones of intense alteration and abundant veining. A study of fault structures may reveal a fracture pattern that should result in strong disturbance of a selected area. Within this area abundant veining by quartz, calcite, ankerite, barite, and/or siderite gives added evidence of structurally developed openings reaching downward toward the source of solutions and gases normally expected to carry ore minerals toward the surface. The degree of mineralization by the sulphides at shallow horizons in these veins is not a measure of the probable productiveness of deep-lying ore shoots, but the presence of these potential ore shoots is indicated variously by the shallow presence of some of the sulphides sought. The author believes that careful consideration of mapped areas of strong structural conditions, alteration, and veining will reveal sufficient evidence to allow careful projection of the features downward toward a potential locus of mineralization. The deeper development program should be directed toward this indicated locus, toward anticipated favorable wall rock conditions, and toward the zone within which siderite may be expected to be the prevailing gangue in contrast with the lighter-colored carbonates previously mentioned.

As a generalization, it may be said that many almost barren veins in the shaly rocks of the Wallace formation contain rich commercial ore shoots shortly after entering the more brittle friable quartzite of the underlying St. Regis formation. Examples of this are the Sunshine, Polaris, and Rambo-Omega-Rotbart areas in the Chester vein, and the Galena vein opened in the Vulcan shaft development. There are several other situations where this approach may have a good chance of success.

Although it is very often possible to predict the position of the top of more brittle bleached rocks with a fair degree of certainty, it is difficult to predetermine the zone within which other carbonates may give way to siderite, the most favorable gangue material known thus far.

The Polaris mine adjoins the Sunshine mine on the east, and the original claims covering the vein outcrops were among the earliest locations in the Silver Belt. Siderite float was explored by shallow pits, and minor silver-copper mineralization influenced the early day operators to drive the upper tunnel. Drifting on the vein structure was encouraged almost continuously by the presence of some sulphides, and near the westerly end of the tunnel a narrow streak of tetrahedrite in a fairly strong siderite vein was stoped for a length of about 90 ft and for 30 to 60 ft above the level. Winze exploration 70 ft below the level failed to find commercially significant ore, and the project lay dormant until the Hecla Mining Co. became interested in 1930. The situation on the tunnel level at that time is shown in the upper part of Fig. 3.

Although the approach to deeper development was influenced to a great extent by the requirements of shaft location, lack of knowledge of the environment of Silver Belt ore shoots and property situations, the Polaris orebody was discovered on the 920 level as shown in Fig. 3. A large amount of exploration was done on unproductive veins east and north of the ore shoot before commercial ore was found. The ore shoot was mined to a point above the 670 level, and subsequent exploration on deeper levels was successful in finding the dip extension.

If the initial objective of the crosscut from the bottom of the shaft had been the downward projection of the locus of veining and mineralization as it was exposed on the tunnel level, the crosscut would have found the orebody, and much of the other exploratory work would have been unnecessary.

Early in the Polaris development program the author made use of the system of vein contouring described by H. J. C. Conolly[9] in 1936. In the deeper exploration the trend lines so developed indicated the down-dip projections of important ore shoots, and this method applied to the adjoining Sunshine mine, in conjunction with the Polaris, also gives an excellent demonstration of the adaptability of the method. The Polaris veins are being mined on Sunshine's 2700 and 3100 levels, respectively 10 ft and 400 ft below sea level.

The Omega-Rotbart veins are located easterly from the main part of the Sunshine mine, on the north or footwall side of the Silver Syndicate fault, and on the hangingwall side of an expression of the Chester fault, see Fig. 4. In the development of the Polaris vein via the 1000 level, or Polaris lower tunnel level, the fault-vein relationships were found as illustrated. A strong structural environment, abundant veining, and minor amounts of the sulphides were well indicated on this level. Development of the projected locus of mineralization on the Polaris 2300 level gave further encouragement for the deeper development from the easterly end of the Sunshine mine on Sunshine 2700 level where bonanza silver-lead ore was opened. The 3100 level is shown in Fig. 4 where a length of almost 1000 ft of continuous high-grade silver and silver-lead ore was developed. This ore shoot has been opened through a vertical range of more than 1400 ft reach-

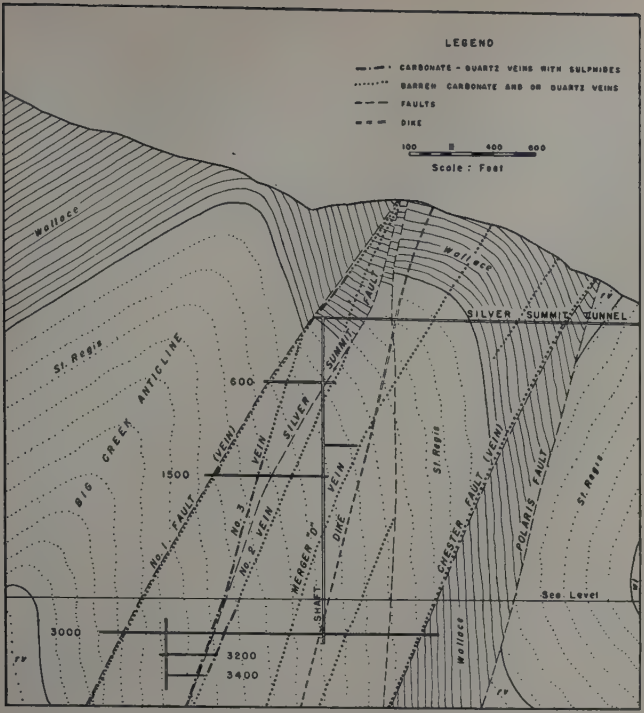


Fig. 6—Section through Silver Summit mine looking westerly.

ing from above the Sunshine 2300 level to the 3700 level, the latter level being about 1000 ft below sea level.

The Silver Summit mine, recently added to the list of producers of silver ore in the Silver Belt, has been described by the author in earlier papers.[6,7] It is located almost a mile easterly from the east end of the Rotbart area described above. On the upper tunnel level early day prospectors found small veins as shown on Fig. 5, and later exploration by Harry Pearson and associates on the lower tunnel level found the veins and fault structures shown on the center portion of Fig. 5. Some of the veins were entirely barren quartz or quartz and carbonate veins containing carbonate of light color and others contained some siderite. The No. 1 vein was largely siderite with minor amounts of tetrahedrite and pyrite, and in one locality the vein reached a width of almost 14 ft of solid siderite. A shaft was sunk in successive stages to the 600, 1200, and 1500 levels, and some work was done on the most southerly vein structure on 600 and 1500 levels without success. Polaris Mining Co. financed the development from the 1500 level to the 3000 level. On the new bottom level the direction of the south crosscut was determined to a great extent by the position of various development objectives planned for the level. The development program found the Silver Summit ore shoots as they are shown on Fig. 5 and partially described in a former paper.[7]

On the lower tunnel level the structural environment was strong, bleaching was pronounced, veining was prolific, and the sulphides sought were clearly in evidence in several of the veins. The down-dip projection of these structures, which contained evidence that they were tributary to the locus of mineralization, point directly toward the ore shoots found on the deeper levels.

Since the vein was opened on the 3000 level an offset winze was sunk, and the 3200 and 3400 levels opened the ore shoots. Mining is proceeding on all levels, and during 1950 net smelter returns totaled $1,251,173.

Fig. 6 shows some of the relationships of folding, faulting, and veining through the Silver Summit shaft.

Other mines in the Silver Belt, including the main Sunshine, the Coeur d'Alene mines, and the Galena or Vulcan, appear to exhibit the characteristics described in this paper. Several prospects including Silver Dollar (Lincoln), Nellie, Argentine, Rock Creek and possibly others appear to exhibit in varying degrees the above characteristics.

Although the principal objective of this paper is to point out some possible shallow expressions of Silver Belt ore shoots, the following features also have been observed to exert a local control over the location of ore shoots:

1—Promising veins where they cross folds or warps in the wall rock.

2—Warps in veins both on strike and dip (see contouring method references above).

3—Where veins approach prominent fault structures.

4—Where strong veins in unbleached or partially bleached rock enter zones of intense bleaching.

5—Where veins in argillaceous rocks enter more brittle quartzitic rocks.

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"Depletion" in Federal Income Taxation of Mines

by K. S. Benson

DEPLETION is a subject of vital importance to the mining industry. Yet, in spite of its importance, its significance is not generally understood. The purpose of this discussion is to clarify the main aspects of the subject from the viewpoint of a metal mine taxpayer.

To define the term depletion, it is necessary to distinguish among its various uses. In the economic or geological sense, depletion means the exhaustion of a natural resource. A mineral deposit is a wasting asset and once exhausted is nonrenewable. Millions of years were needed to produce an ore deposit, which may be consumed in a few years and which cannot be replaced except by the discovery of new sources of supply. The wasting asset feature of the mining industry has a vital bearing on the impact of the Federal Income Tax Law on this industry. This is recognized in the law by the various provisions dealing with the depletion allowance, and in this sense the term depletion has an income tax meaning.

Depletion from the tax viewpoint means the statutory deduction from gross income designed to permit the return to the taxpayer of the capital consumed in the production and sale of a natural resource. The mining enterprise realizes income on the extraction and sale of minerals and a portion of the income realized represents capital consumed. As the resource is exhausted, the mining enterprise approaches the end of its existence unless new sources of supply can be acquired. Depletion from the tax viewpoint is a creature of statute with limited meaning and application and, in essence, is a method for amortizing the value of the primary asset of a mining enterprise.

An example can best illustrate the significance of depletion from the tax viewpoint. Compare a manufacturing concern with a mining company. In computing taxable income of a manufacturing concern, consideraion is given to the cost of producing such income, the principal costs being capital investment for plant and equipment, labor, and raw materials going into the products produced. A mining enterprise, on the other hand, is faced with a different problem because its principal asset is the natural resource which it is producing. In computing its taxable income, consideration is given also to its capital investment for plant and equipment and the cost of labor; but in addition, recognition must be given to the fact that a portion of the proceeds realized on the sale of mineral represents capital. Without such recognition, the mining company would be taxed not on income but on capital and income, and Congress has never intended that capital be taxed as income.

Thus, when depletion allowable is referred to in the mining industry, it means the statutory deduction allowable in computing taxable income of a mining enterprise. For guidance the appropriate provisions of the Internal Revenue Code, Income Tax Regulations, and the judicial decisions interpreting and construing them must be examined. It is important to identify and distinguish three methods of determining the allowance for depletion: 1—Cost depletion, 2—Discovery depletion, and 3—Percentage depletion. The basic method is cost depletion and in addition some taxpayers may be entitled to use discovery depletion and other taxpayers may be entitled to use percentage depletion. Discovery depletion and percentage depletion, however, are mutually exclusive and if a taxpayer is entitled to percentage depletion, he is not entitled to discovery depletion. By statute, a metal mine taxpayer is entitled to use cost depletion or percentage depletion, whichever produces the highest deduction. Thus, discovery depletion is merely of academic interest to such taxpayers and to most others. Briefly and broadly speaking, these methods of determining depletion may be described as follows:

1—*Cost Depletion*: Under this method, the allowable deduction for depletion is based upon the cost of the particular deposit to the taxpayer, unless the deposit was owned prior to Mar. 1, 1913, in which case the taxpayer may use the fair market value of the deposit on that date or actual cost, whichever is higher. This method is sometimes described as *basis depletion* or *adjusted basis depletion*, but in this discussion it will be referred to as *cost depletion*.

2—*Discovery Depletion*: Under this method, the allowable deduction for depletion is based on the fair market value of the deposit at the date of discovery or within 30 days thereafter and was originally designed to take into account deposits discovered subsequent to Feb. 28, 1913.

3—*Percentage Depletion*: Under this method, the allowable deduction for depletion is based on a specified percentage of the income realized during the taxable year from a particular property.

As stated, the concept of depletion is based upon the exhaustion of a natural resource as distinguished, for example, from the concept of depreciation based on the exhaustion of property used in trade or business. From the tax viewpoint, depletion first became important in the administration of the Corporation Tax Act of 1909, which provided for an excise tax on net income. As soon as this act went into effect, mining taxpayers attempted to claim a deduction for depletion in computing net income although there was no specific mention of a deduction for depletion in the statute. The courts in these cases uniformly held that the statute did not permit an allowance for depletion in computing net income and also held that the provision permitting a reasonable allowance for depreciation did not include depletion. These early cases are quite significant because they establish the principle that the

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deduction for depletion is not a matter of right but of Congressional grace, and reference must be made to specific statutes to determine the scope of the depletion deduction.

As a result of the hardship to operators of mines, Congress in the first income tax law (the Revenue Act of 1913) provided for ". . . a reasonable allowance for the exhaustion, wear and tear of property arising out of its use or employment in the business, not to exceed, in the case of mines 5 pct of the gross value at the mine of the output for the year for which the computation is made, . . .". The word *depletion* first made its appearance in the 1916 Revenue Act, which provided for a reasonable allowance for depletion in the case of mines not to exceed the market value in the mine of the product thereof mined and sold during the year for which the return and computation are made. The 1916 Act limited the deduction for depletion to the recovery of cost, or in the case of deposits acquired prior to Mar. 1, 1913, to the fair market value as of that date. The principal change in the 1918 Revenue Act was the introduction for the first time of discovery depletion in the case of mines, oil, and gas wells discovered by the taxpayer on or after Mar. 1, 1913, with the depletion allowance based upon the fair market value of the property at the date of discovery or within 30 days thereafter.

Essentially two methods of computing depletion had developed, namely, cost and discovery depletion. During the ensuing years, it became evident that the administration of both cost depletion and discovery depletion was too complex and many inequities were unavoidable, so that the remedy seemed to be the adoption by Congress of a different, simpler and fairer method of calculating the allowance for depletion. In the 1926 Revenue Act, Congress, in the case of oil and gas wells, adopted percentage depletion so as to permit the deduction for depletion to be based upon a percentage of gross income derived from the properties, limited to 50 pct of the net income of the taxpayer from each property, and at the same time eliminated discovery depletion with respect to these items. In the 1932 Revenue Act, percentage depletion was extended to coal, metal mines and sulphur, and at the same time discovery depletion eliminated with respect to these items. Since that time, the benefits of percentage depletion have been extended to many other items.

This summarization, while not complete, points out the manner in which the methods of computing depletion have developed, and illustrates that once Congress recognized the concept of depletion for tax purposes the main problem was, and still is, the determination of the amount and method of computing the depletion deduction.

Sections 23(m) and 114(b) of the present Internal Revenue Code relate to the allowance for depletion and reference must be made to these sections as well as to the regulations issued by the Commissioner to determine the scope and application of the present depletion provisions. Section 23(m) provides, in computing net income, there shall be allowed as a deduction, "In the case of mines . . . a reasonable allowance for depletion and for depreciation of improvements, according to the peculiar conditions in each case; such reasonable allowance in all cases to be made under rules and regulations to be prescribed by the Commissioner, with the approval of the Secretary." Section 114(b) outlines the methods of calculating depletion and under this section of the Code a metal mine taxpayer may be entitled to a deduction for depletion based upon the cost method or upon the percentage method, whichever is higher.

The language of Section 23(m) clearly points out that depletion is not restricted by the form of legal entity seeking depletion. Thus it is allowable to individuals, corporations, partnerships, trusts, etc., assuming, of course, that the other requirements of the Code and regulations are met. All taxpayers, however, who seek a deduction for depletion must be the owner of an economic interest in a mineral deposit. Treasury Regulation 111, Section 29.23(m)-1 defines an economic interest as being possessed, ". . . in every case in which the taxpayer has acquired, by investment, any interest in mineral in place . . . and secures, by any form of legal relationship, income derived from the severance and sale of the mineral . . ., to which he must look for a return of his capital. But a person who has no capital investment in the mineral deposit . . . does not possess an economic interest merely because, through a contractual relation to the owner, he possesses a mere economic advantage derived from production." This requirement that a taxpayer be the owner of an economic interest has caused much litigation, and the definition set forth in the regulation developed over a period of many years.

In addition to determining whether a taxpayer is eligible for the depletion allowance, the economic interest test is important in determining the tax consequence of certain transactions. For example, a taxpayer may claim that he has sold his interest in a mineral deposit to obtain a capital gain, but if he has retained an economic interest in the mineral, the payments received are ordinary income subject to depletion. On the other hand, a taxpayer may claim a deduction for depletion only to find that he has made a *sale* and thus no deduction for depletion is allowable. The allowance of depletion and the benefit of the capital gain provisions are mutually exclusive.

The test of economic interest has been, and still is, difficult to apply in determining whether or not the taxpayer is entitled to a deduction for depletion. At one extreme, it is clear that the sole owner and operator of a mineral property is entitled to depletion because he, of all persons, clearly has an economic interest in mineral in place. At the other extreme, a person who sells his interest in a mineral is not entitled to depletion because he has converted his economic interest in mineral in place into another form of capital. In between these two extremes, there are a myriad of transactions which fall on one side or the other.

1—*Royalty Payments:* In the usual lessor-lessee relationship, a mineral property is leased by the owner to an operator. The lessor receives a royalty rate from the lessee, and the lessee, realizes income from the production and sale of the mineral. Both the lessor and lessee have an economic interest and both are entitled to depletion. The lessor is entitled to depletion on income represented by royalty payments received, and the lessee is entitled to depletion on income realized on the extraction and sale of the mineral. It is immaterial whether the royalty is payable in mineral, or in a sum of money per unit extracted, or as a percentage of gross profits, or as a percentage of net profits. It should be noted, however, that when royalty is based upon a percentage of net profits some question may still be raised as to whether the lessor is entitled to depletion. For many years it was held by the courts that a share in the net profits from the development and operation of a mineral property does not entitle the holder of such an interest to a depletion allowance even though continued production is essential to the realization of such profits. In two recent cases involving oil leases (Kirby Petroleum Co. vs. Commissioner 326 U.S. 599 and Burton-Sutton Oil Co., Inc. vs. Commissioner 328 U.S. 25) the Supreme Court ruled that a lessor receiving a share of the net profits realized by the lessee from operations under a lease is entitled to depletion on such share of the net profits because this interest constitutes an *economic interest.* The rule of these decisions has been followed in other recent cases and it is probable that this rule will be applicable to other types of leases.

2—*Stock Interest:* A stock interest does not constitute an *economic interest* in mineral in place. Therefore, a stockholder of a mining company or an oil company receiving dividends is not entitled to depletion upon such dividends. However, in some instances if the company making the distribution does not have any earnings or profits but makes the distribution from reserves arising by reason of cost depletion, the dividend may be nontaxable as a return of capital. Likewise if a stockholder in a mining company or an oil company sells his stock under an arrangement whereby he is to share in the net profits from the development and operation of the property of the oil company or mining company, such taxpayer is not entitled to a depletion deduction against the payments received under this arrangement.

3—*Stripping Contracts:* Although it does not appear that any stripping contract cases have developed involving metal mines, the problem has arisen in the case of coal mines. The Commissioner has ruled that contractors who have undertaken to remove overburden and mine coal may, dependent upon their rights in respect of the properties involved, be entitled to the deduction for depletion as in the case of other mineral properties. The allowance is warranted only where under the agreement between the parties, the stripping contractor obtains a capital interest in the mineral in place and must look to severance and sale of the mineral for the return of capital consumed in the process. Thus, if the compensation of the contractor is, by the terms of the contract, to be derived solely from the sale of the coal, depletion is allowable. If, on the other hand, the compensation of the contractor is dependent entirely upon removal of the overburden and other services with no part of the compensation derived from the extraction and sale of the coal, the deduction for depletion is not allowable. In some cases, it may be necessary to split the income received by the contractor so that the income derived from the mining and sale of the coal would be subject to the depletion allowance.

4—*Cash Payments:* If a lease is sold for cash plus a royalty, the seller-lessor is entitled to depletion allowance on only that part of the income arising from the royalty payment. If, however, the cash payment is an advance royalty, and the presence of an overriding royalty usually makes the cash payment an advance royalty, depletion is allowable on not only the royalty income but on the cash payment income. In connection with advance royalty and minimum royalty, Treasury Regulation 111, Section 29.23 (m)-10 prescribes a rule which may seriously affect a lessor. For example, if a lessor receives advance royalty or minimum royalty under the terms of a lease prior to production, he is entitled to a depletion allowance on such income in the year of receipt in accordance with the terms of the aforementioned regulation but if the mineral rights granted expire, terminate or are abandoned before the mineral which has been paid for in advance has been mined, the grantor must report in the year of such expiration, termination or abandonment income equal to the depletion deductions of prior years, and such prior deductions are restored to his capital account.

In addition to meeting the requirement of economic interest, the metal mine taxpayer must qualify as an operator of a *metal mine.* In this connection both the terms *metal* and *mine* have technical meanings. If low grade chromite ore is mined and pressed into refractory brick, no depletion is allowable because the mineral is not being used for a metal purpose. If the mineral, after mining, is processed into chromium metal, depletion would be allowable. Thus it is evident that the so-called *use test* can cause many complications. As to the meaning of the term *mine,* the best illustrations of the problem appear in the so-called tailings dump and lean ore cases that are becoming increasingly more important.

Income may be derived from reworking a tailings dump or from working a low grade ore dump, and this in turn raises the question of whether a deduction for depletion is allowable against such income. In the cases involving this point, the decisions have generally turned on whether a tailings dump or a lean ore dump is a mine within the meaning of Section 23(m). In the Kohinoor Coal Co. case (171 F. (2d) 880 (CCA 3rd) it was held that the lessee of a culm or refuse bank, thrown aside by the owner-operator of a coal mine, was not entitled to percentage depletion, and this case followed the rule of Chicago Mines Co. vs. Commissioner (164 F. (2d) 785 (CCA 10th)) and Consolidated Chollar Gould & Savage Mining Co. (133 F. (2d) 440 (CCA 9th). In this latter case, the taxpayer extracted ore from tailings dumps composed of broken ore bearing rock taken from adjoining mines, and the court held that such extraction did not constitute operation of a mine and percentage depletions was not allowable. On the other hand, the Ninth Circuit Court of Appeals has held that a taxpayer is entitled to depletion on income derived from reworking a tailings dump previously created by him by processing ore from his mine (Commissioner vs. Kennedy Mining & Milling Co., 125 F. (2d) 399) and has also allowed depletion on income derived by a lessee of a mine from the treatment of tailings left by a previous operator (New Idria Quicksilver Mining Co. vs. Commissioner 144 F. (2d) 918). While attempts have been made to distinguish these various cases, in all probability the question will never be free from doubt until ruled upon by the United States Supreme Court or until clarified by Congress.

There are several methods of computing the depletion allowance, assuming a taxpayer is entitled to the deduction for depletion. As previously pointed out, in computing the allowance for depletion, a metal mine taxpayer should make two calculations, namely, the depletion allowable by the cost method and depletion allowable by the percentage method because he is entitled to the greater deduction.

Cost Depletion: To determine the allowance for cost depletion in a particular year, the taxpayer must establish three factors: 1—The basis upon which the depletion allowance is to be computed, 2—The estimated total recoverable units of mineral in the property, 3—The number of units of mineral sold in the year. Expressed in another way, the taxpayer divides his adjusted basis for the property by the total remaining units of mineral in the property to obtain the depletion unit. Then he multiplies the depletion unit by the number of units of mineral mined and sold in the year in question.

There are many technical questions involved in the determination of the basis upon which depletion is to be allowed, and each taxpayer should read carefully the provisions of Section 113 of the Code, along with the pertinent Treasury Regulations and Court decisions. Simply stated, basis means a money value assigned to property for tax purposes, and in the case of cost depletion it is the money value assigned to a mineral property for the purpose of computing the depletion allowance. Normally this is cost unless the property was acquired prior to Mar. 1, 1913, in which case it is cost or the fair market value of the property as of that date, whichever is higher, properly adjusted in accordance with the provisions of Section 113. Once the adjusted basis has been obtained, it normally is the basis for the entire property, and for depletion purposes this must be further adjusted so as to eliminate any cost or value of the land for purposes other than mineral production, to eliminate any amounts recoverable through depreciation and from deductions other than depletion, and to eliminate the residual values of such property at the end of the operation. Once the basis for depletion has been determined, it remains the same during the life of the property except that it is reduced by depletion deductions allowed and increased by allowable capital additions. On this latter point, Regulation 111, Section 29.23 (m)-15 provides that any expenditures in excess of net receipts for minerals sold shall be charged to the capital account recoverable through depletion while the mine is in the development stage. A mine is considered to have passed from a development to a producing status when the major portion of the mineral production is obtained from workings other than those opened for the purpose of development; or when the principal activity of the mine becomes the production of developed ore rather than the development of additional ore for mining. Thus, for example, in an open pit iron ore mine, the stripping cost prior to the time the mine becomes a producing mine is capitalized and recoverable through depletion, and the amount expended for this pre-production development would be an allowable capital addition to the basis for the calculation of cost depletion. After the mine reaches the producing stage, any stripping costs would be capitalized and deducted ratably against the production benefitted. Such deductions would be in addition to the allowance for depletion. On the other hand, whether in the pre-development or post-development stage, expenditures for plant and equipment and for replacements, not including expenditures for maintenance and ordinary and necessary repairs, shall ordinarily be charged to the capital account recoverable through depreciation.

After determining the basis for depletion, the next step is to determine the number of total recoverable units of the mineral in the deposit in the year in question. This involves an estimate on the basis of reasonably known facts at the time the estimate is made. The estimate is to be in accordance with methods current in the industry and the problem is primarily an engineering one. In the event either the Commissioner or the taxpayer, as a result of operations or development work, determines that the total recoverable mineral units are either greater or less than the prior estimate, a revision thereof is made and the revised estimate is used in determining the depletion unit.

Once the basis and total estimated units of mineral have been determined the depletion unit can be arrived at, and this unit is then multiplied by the units of mineral mined and sold within the taxable year. A simple illustration will point out the mechanics of computing cost depletion, although in practice most computations are not quite as simple. It may be assumed that a taxpayer purchases the fee of a property for the sum of $480,000 and that this property has no value except for the mineral contained therein. The basis for the purpose of computing cost depletion will be the purchase price of $480,000. If pre-production development expenditures total $120,000, this amount is an allowable capital addition and will increase the basis to $600,000. Assuming that the property contains an estimated 2,000,000 tons of recoverable iron ore, the depletion unit will be 30¢ per ton and on the production and sale of 50,000 tons, the cost depletion allowable would be $15,000 for that year. In the following year, $15,000 would be subtracted from the original basis, leaving a remaining basis of $585,000. If the total recoverable units of mineral remain unchanged (except for the subtraction of prior year's production sold), the depletion unit will continue to be 30¢ per ton until the total basis is recovered. In the event a revision is made of the total estimated units of mineral, the depletion unit will be either larger or smaller depending upon the revised estimate, but in no event can more than the remaining basis of $585,000 be recovered by cost depletion, or an overall total of $600,000 on the facts assumed.

Having calculated the allowance for cost depletion, the taxpayer should next compute the percentage depletion allowable.

Percentage Depletion: In the case of metal mines, the allowance for percentage depletion is 15 pct of the gross income from the property during the taxable year, excluding from such gross income an amount equal to any rents or royalties paid or incurred by the taxpayer in respect to the property but such allowance cannot exceed 50 pct of the net income of the taxpayer (computed without allowance for depletion) from the property. Thus, there are two limitations in the calculation of the percentage depletion allowance, and in some instances the allowance will be determined by the 50 pct of net income limitation, and in other instances the 15 pct of gross income limitation will govern. In view of these limitations, there has been much litigation because the Commissioner has attempted to minimize as much as possible *gross income* and *net income* in order to reduce the allowance. Most problems involving the calculation of percentage depletion center about the meaning of *gross income, net income* and *property*.

Gross income from the property in the case of a crude mineral product other than oil and gas is defined to mean the gross income from mining. The term *mining* includes not only the extraction of the ores or minerals from the ground, but also the ordinary treatment processes normally applied by mine owners or operators to obtain the commercially marketable mineral product or products and so much of the transportation of ores or minerals (whether or not by common carrier) from the point of extraction from the ground to the plants or mills in which the ordinary treatment processes are applied thereto as is not in excess of 50 miles. If the transportation to the plants or mills exceeds 50 miles, the Commissioner must approve the excess transportation if it is to be included in the definition of gross income from the property. In the case of iron ore, the ordinary treatment processes include sorting, concentrating and sintering to bring to shipping grade and form and loading for shipment. If the taxpayer sells his mineral product in the immediate vicinity of the mine, gross income would be the amount for which the product was sold. If, however, the product is transported or processed other than for ordinary treatment processes, gross income would be the representative market or field price without such transportation and processing. In the case of iron ore, for example, the gross income from the property would be the f.o.b. mine value realized by the taxpayer on shipments less any rents or royalties paid or incurred with respect to the property.

Net income of the taxpayer (computed without allowance for depletion) from the property means the gross income from the property less the allowable deductions attributable to the mineral property upon which the depletion is claimed and the allowable deductions attributable to ordinary treatment processes insofar as they relate to the product of such property, including overhead and operating expenses, development costs properly charged to expense, depreciation, taxes, losses sustained, etc. but excluding any allowance for depletion. Deductions not directly attributable to a particular property or process are to be fairly allocated. The net effect of these requirements is to require in the computation of net income of the taxpayer from the property a deduction for almost every item which would be deductible in computing net taxable income from the property except depletion. For example, interest on borrowed money and amounts paid in settlement of silicosis claims are deductible but contributions to charity are not required to be deducted if they are not attributable directly to the mineral property.

Property is defined as the interest owned by the taxpayer in any mineral property, but in the event the taxpayer has interest in separate mineral properties, each interest is a separate property. Where two or more mineral properties are included in a single tract or parcel of land, the taxpayer's interest may be considered a single property, provided such treatment is consistently followed. This definition has caused confusion because one mine may involve two leases or may involve a combination of a fee interest and a lease interest; and for the purpose of percentage depletion, the Commissioner may require these interests to be treated as separate properties, causing difficult problems of allocation.

To illustrate the application of percentage depletion, the example in the discussion of cost depletion may be used. In that example the taxpayer produced and sold 50,000 tons of iron ore with, let it be assumed, an f.o.b. mine value of $5 per ton. No royalty payments are involved because the taxpayer owns the fee of the land and, therefore, his gross income from the property would be $250,000 and 15 pct of the gross income amounts to $37,500. If it is assumed that the net income of the taxpayer from the property, without allowance for depletion, amounts to $1 per ton, 50 pct of the net income on 50,000 tons would amount to $25,000 and the percentage depletion allowable would thus be governed by 50 pct of the net income and would be in the amount of $25,000. In this particular example, the excess of percentage depletion over cost depletion amounts to $10,000. The taxpayer would be entitled to a total depletion deduction of $25,000, and he would adjust his basis for the property by this amount, i.e. it would be reduced to $575,000 for the purpose of calculating cost depletion in the next year. As soon as the basis is fully recovered, the taxpayer will no longer be entitled to cost depletion but may still obtain percentage depletion.

This discussion has touched in a broad way some of the problems involving depletion. In using the term depletion, its meaning from an economic and geological viewpoint and its meaning from a Federal Income Tax viewpoint must be distinguished. Depletion from the tax viewpoint refers to a statutory deduction allowable in computing the taxable income of a mining enterprise. Every Revenue Act since 1913 has contained provisions relating to the amount and method of calculating the allowance for depletion. A metal mine taxpayer who otherwise meets the eligibility requirements is entitled to an allowance for depletion computed by the cost method or by the percentage method, whichever is higher. The determination of the depletion deduction by these methods is largely a mechanical process once the basic factors are established.

The mining industry differs from other industries because the mineral producer is really engaged in two interrelated activities. One activity involves the rendition of the services of mining and processing of minerals to produce a marketable product, and, in this respect, the mining industry is comparable to many other industries. The second activity, however, distinguishes the mining industry because it involves the sale of the mineral producer's supply of mineral. This supply of mineral is the basic capital asset of any mining enterprise, and the production and sale of the mineral ultimately involves the liquidation of the enterprise unless new deposits are discovered and developed. The discovery of new deposits in turn involves the expenditure of large sums of money for exploration, and it is well known that for every successful exploration there are many, many failures. The depletion provisions of the Code are intended to eliminate what would otherwise be an inequity because they recognize that capital should not be taxed as income. Also, these provisions, while in need of some changes to encourage and accelerate exploration and development, do assist in stimulating new risk capital to enter into the mining field and provide a method whereby existing mining enterprises may set aside funds for further exploration.

Metallurgical Applications of the DorrClone

by Frank T. Weems

The basic operating properties of the DorrClone are discussed and certain metallurgical applications which exploit these properties are presented. An effective method of controlling the consistency of the DorrClone underflow is described.

THIS paper presents in a general manner some of the applications of the DorrClone (Dutch States Mines cyclone) in the metallurgical and heavy chemical fields. Technical or mathematical analyses of the design factors and of the mechanism whereby the cyclone effects its separation have been capably treated in a series of previous papers.[1-4] Engineering and operational aspects of a few installations and laboratory tests have been chosen to emphasize the unique classification characteristics of the DorrClone and to illustrate its utility in certain difficult situations. Its use in coal washing and heavy-media operations are not considered.

Some 25 or more industrial installations and applications of the DorrClone in the metallurgical fields are operating at present, ranging from single trial units to complete batteries composed of a multiplicity of units. The data were obtained in part from some of these, and in part from operations of units at the Westport Mill, the laboratories and testing plant of the Dorr Co. at Westport, Conn.

The DorrClone was developed, in principle, by the research dept. of the State Mines, Limburg, Holland, during World War II. Its first use was to dewater a —50 micron sand used as a dense medium in coal washing. The metallurgical wet cyclone was first publicized by Driessen[5] and Yancey and Geer.[6] Since then the cyclone has been studied as a means of dewatering, heavy media concentration, and fine size classification.

The DorrClone is shown in Figs. 1 and 2. Pulp to be treated is pumped through the feed nozzle, A, into the feed cylinder, B. The feed nozzle is of rectangular cross-section, the outermost edge being tangent to the feed cylinder walls. Feed shims, C, may be attached to the innermost walls to adjust the open area of the feed nozzle available to entering pulp.

The tangential injection of feed under pressure supports the vortex action of the pulp that has been detained in the cyclone; the centrifugal forces of this vortex throw the coarse particles to the walls of the cone, D, where they collect and pass downward and out through the adjustable apex opening, E. The fine particles contained in most of the water that entered with the feed pulp move to the inner spiral of the vortex and flow out through the vortex finder, F, contained in the orifice plate, G, which forms the top of cyclone chamber. The overflow pulp may then spray freely into a spray chamber, H, surmounting the cyclone or pass directly into a 90° elbow, I, of somewhat larger diameter than the vortex finder.

Installation

Where more than one cyclone is to be installed, best overall efficiency and ease of control are obtained with equal distribution of both pulp volume and weight of solids among the cyclones, and with the same pressure being maintained at each cyclone feed nozzle. Fig. 3 is a suggested arrangement for feeding two or more cyclones in parallel.

Feed pressure control: Under normal operating conditions it is not unusual for some surge to exist in the feed that is to be sent to the cyclone. As will be discussed later, surges in the solids content of the pulp are usually not of great operational consequence, but surges in feed volume are a problem.

Several systems have been devised for absorbing surges in feed volume, each involving a surge tank. If the surges are small and fairly rhythmical, a tank of sufficient size might be installed to dampen out

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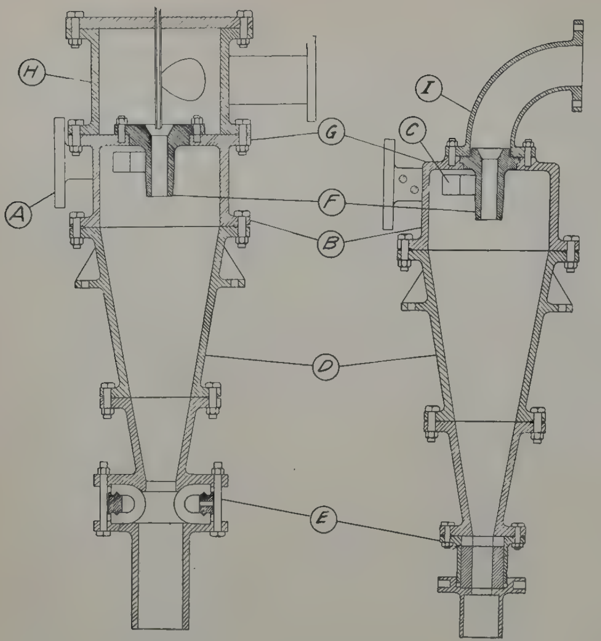

Fig. 1—The DorrClone.

A — Feed nozzle
B — Feed cylinder
C — Feed shims
D — Cone
E — Adjustable apex opening
F — Vortex finder
G — Orifice plate
H — Spray chamber
I — Elbow

the surge. If the surges are large or of long period, the pump speed should be changed by some control or regulator system,[7] and the number of cyclones on the line should be varied to keep the inlet pressure relatively close to the optimum value. For conditions between these two extremes, one of two methods might be used: 1—install pumps of slightly greater capacity than required for the maximum surge and then add water to keep a constant level in the feed tank, or 2—recycle overflow as shown in Fig. 4. Where a clean underflow is the prime requisite, method 1 is recommended; for maximum recovery of solids in the underflow, method 2.

DorrClone Underflow Control

If full advantage is to be taken of the ability of the DorrClone to produce a dense underflow, some method must be adopted whereby physical adjustments of the cyclone are made automatically in response to the normal operating variations in feed composition. The density of the underflow from the cyclone has been shown to be a function of the apex orifice diameter if the feed rate and analysis are fixed[2, 3, 6, 8, 9]; and, as a corollary, the density of the underflow may be controlled by adjusting the apex orifice opening as the feed rate and composition vary. By decreasing the apex orifice opening, the density of the underflow is increased.

The adjustment of the apex orifice to control underflow density should be made in a manner that will minimize disturbance of the normal flow pattern of the cyclone. The most convenient device now known is an adjustable aperture.

There are two types of restricted apertures that have been used with good results: 1—A rubber doughnut that may be compressed mechanically to close its inner diameter, shown in Fig. 1. 2—A rubber tire that may be inflated to close its inner diameter. Another type of closure consists of a cone that can be inserted or withdrawn from a fixed apex opening to decrease or increase the area available for pulp discharge. This gives underflows of somewhat lower density.

It has been found experimentally that a vacuum exists in the core of an operating cyclone. The magnitude of this vacuum varies with the consistency of the underflow pulp, other operating factors being fixed. The absolute value of the vacuum corresponding to some given underflow consistency will be a function of such factors as feed pressure, physical dimensions of the component parts of the DorrClone, length and size of the overflow pipe, etc. However, having a cyclone operating under conditions of normal variations, the vacuum in the air core of the cyclone can be correlated with the density of the cyclone underflow. In the region of underflow densities commonly desired, this vacuum will change from a few inches to several feet of water vacuum as the underflow density changes only a small percentage. This vacuum has been used effectively as the intelligence in controlling the consistency of the DorrClone underflow.

Work on the automatic control system, based on vacuum, was initiated by Weaver and Wright of the American Cyanamid Co. and subsequently developed in its present aspects, see Fig. 5, by Fitch of the Dorr Co.

The DorrClone can be equipped with any type of sensitive nonhunting control instrument capable of operating in response to a 0 to 10 or 0 to 20 in. of water control range. It is preferable to have manual adjustment for both proportional band and reset time. The intelligence is applied to this instrument, the output of which normally operates a pressure booster. A 100-psi compressor is connected through this booster to the apex valve, thereby allowing the pressure in the apex valve to be varied between zero and about 85 or 90 psi in response to changes in output pressure of the controller. The vacuum at the probe, therefore, controls the pressure in the apex valve. This pressure determines the size of the apex orifice, which in turn controls the density of the underflow.

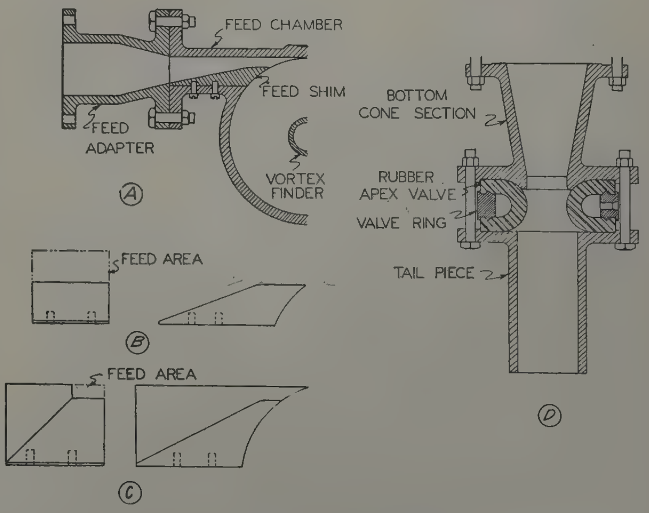

Fig. 2—Details of DorrClone.

A — Section through feed nozzle
B — Small shim
C — Large shim
D — Detail of tire-type apex valve

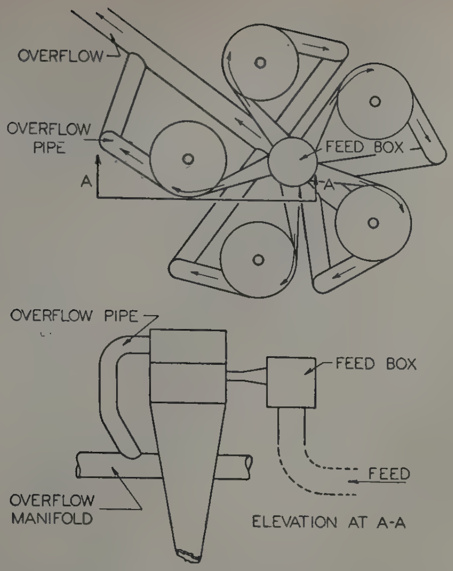

Fig. 3—Multiple DorrClones feed and overflow assembly.

The average consistency of the underflow is set by adjusting the control point of the instrument. Response of the instrument to changes in vacuum is practically instantaneous. The magnitude of the response to a given change in vacuum, however, can be controlled by setting the proportional band and reset time. These adjustments should be balanced to give the closest possible control without the risk of hunting.

As an example, the control is set for 6 in. of water vacuum, corresponding to an underflow density of 72 pct solids and an apex valve pressure of 75 psi. If the feed pulp becomes more heavily loaded with coarse material that reports to the underflow, the underflow volume (hence density) will increase and the probe vacuum will tend to increase to maybe 10 in. vacuum. The control instrument's output will decrease the apex valve pressure, allowing the orifice to open. As the valve opens, the probe vacuum decreases until equilibrium is re-established at 6 in. vacuum. The density of the underflow will again be about 72 pct solids.

By having this instantaneous positive control action, plugging of the underflow by normal changes in feed density is virtually impossible.

A 24-in. DorrClone to be used in desliming Florida phosphate rock at the Homeland Mine of Virginia Carolina Chemical Co. was equipped with the vacuum-type underflow density control system. Details of testing arrangement are given later.

The vacuum developed in the cyclone was taken by a ¼-in. probe extending into the vortex finder to a Foxboro Model 40 Stabilog controller. The controller operated through a 3 to 1 pressure booster to adjust the pressure in a tire-type apex valve.

When using this method of underflow consistency control, the recovery of valuables as well as the density of the underflow show little change with rather large changes in solids content of the feed. The results of a 7-hr test with samples taken at 30-min intervals are shown in Table I and are plotted in Fig. 6. There are rapid fluctuations of the feed between 29.1 pct and 46.0 pct—200 mesh, 7.8 pct and 21.4 pct solids, and equally rapid variations in the overflow between 2.9 pct and 10.1 pct solids. The rake product varied only between 66.7 pct and 72.8 pct solids, the recovery of +150 mesh material varied between 97.9 pct and 99.4 pct recovery in the underflow, and the elimination of slimes varied between 84.1 pct and 86.5 pct of the slimes that entered with the feed reporting to the overflow.

In Fig. 6 it is shown that the underflow density can be held constant within the control limits of ±3 pct, independent of the per cent solids in the feed. The small changes in per cent solids of the underflow do not seem from this plot to be related to either feed or overflow density. Through this control of underflow density, the elimination of slimes and recovery of valuable material are held at a constant high value, as shown in Fig. 6, in spite of the large rapid changes in feed and overflow consistency.

Applications and Limitations

The mechanism whereby the cyclone makes its separation is generally considered as the settling of discrete particles under the influence of centrifugal force. The high shearing stresses in the cyclone tend to break up and prevent the reformation of any flocs that exist in the pulp. At the same time, the high centrifugal forces acting on the particles suspended in the pulp will give sufficient force to even the extremely small particles to enable them to exceed the yield point of the pulp and settle through it. Following are mechanical features of the unit: 1—The elimination of flocs through shearing forces, 2—the increase in the acceleration with which particles act against the restraining force (yield strength) of the pulp, and 3—the reduction of effective viscosity through increased rate of shear. These features enable classification in the cyclone free from the undesirable effects of flocculation, even without dilution or the addition of chemical dispersing agents. This makes the cyclone especially attractive for fine separation with pulps that are naturally flocculent.

The cyclone has produced separations in which the top size of the fine product range from 10 to 150 microns; the exact separation being a strong function of the specific gravity of the solids and the plasticity of the pulp. In any given case the cyclone will not require nearly as much dilution water to achieve the fineness of separation as a hydroseparator. In addition, the underflow from the cyclone may be delivered to the process at only slightly lower per cent solids than the rake product from a classifier and at much higher per cent solids than the normal hydroseparator underflow, even when operating on a thoroughly dispersed pulp. This makes the cyclone useful for desliming ahead

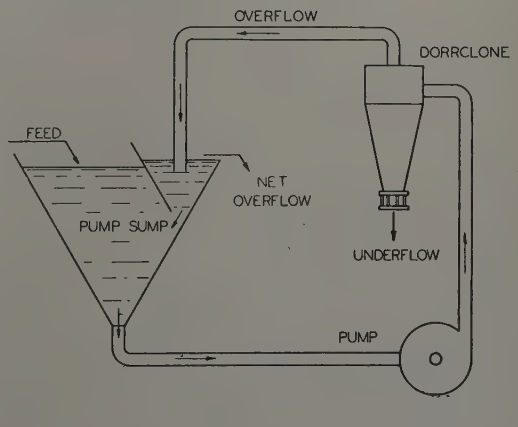

Fig. 4—A suggested feed arrangement.

Table I—Time Samples from Homeland Cyclone

Control Feed Pressure, 14 Psi—Cyclone, 24 In., 20°—Vortex Finder, 6 In.—Feed Entrance, 20 Sq In.—Apex, Rubber Restricted

	Feed				Underflow				Overflow				Recovery, Pct	Recovery, Pct	Removal, Pct
	−200	Solids, Pct	Tons Per Hr	Gal Per Min	−200	Solids, Pct	Per Hr Tons	Per Min Gal	−200	Solids, Pct	Tons Per Hr	Gal Per Min	+200	+150	−200
9:00 am	45.8	20.4	49.3	785	5.6	71.4	27.3	93	95.7	10.1	22.0	692	96.4	99.1	93.2
9:30	37.2	16.2	29.8	776	5.0	71.0	19.3	66	91.2	6.3	10.5	710	98.0	99.4	96.5
10:00	40.0	13.5	27.0	798	5.3	68.1	16.9	62	97.8	4.5	10.1	736	98.6	99.0	91.8
10:30	46.0	16.7	28.2	792	12.0	68.5	17.1	62	98.5	6.9	11.1	728	98.9	99.4	84.1
11:00	40.0	15.5	36.5	822	8.8	68.3	23.6	86	97.4	7.1	12.9	736	98.5	99.2	85.8
11:30	35.0	18.5	37.5	802	4.7	72.0	25.1	83	95.7	6.9	12.4	719	97.8	99.3	91.0
12:00 n	41.6	11.0	27.9	790	5.4	67.9	16.8	62	96.5	5.4	11.1	728	97.6	98.9	92.2
12:30	40.9	11.6	22.5	805	7.4	69.5	14.2	51	98.4	4.4	8.3	754	99.0	99.5	88.5
1:00	43.8	15.0	38.8	778	6.1	71.3	22.7	77	97.2	7.5	16.1	701	97.9	99.2	91.9
1:30	47.1	11.2	23.2	788	10.2	66.7	13.5	52	98.3	5.1	9.7	736	98.6	99.0	87.4
2:00	32.2	16.9	39.9	792	4.0	72.8	27.6	91	95.4	6.9	12.3	701	99.3	97.9	91.5
2:30	29.1	21.4	45.3	778	3.9	72.5	32.0	105	89.8	7.5	13.3	673	95.6	98.2	90.6
3:00	42.4	7.8	19.2	817	4.2	72.7	11.4	37	98.6	2.9	7.8	780	99.0	99.2	94.1
3:30	25.9	18.7	36.6	787	2.1	72.1	26.3	87	87.1	6.3	10.3	700	95.1	97.9	94.3
4:00	36.9	9.0	23.8	801	3.1	72.0	15.3	51	97.1	3.7	8.5	750	98.3	99.0	94.6

Table II—Wear Comparison of Construction Materials

Material	Thickness, In.	Life
Mild steel	¼	125 to 140 hr
Hard alloy steel, (350 Bhn)	¼	6 to 8 weeks
Neoprene	⅜	8+ weeks
Gum rubber	½	in use since March 1950, no wear

of flotation, where a high density conditioning is essential.

Because of the mechanical features of the separation occurring in the DorrClone, clear overflows of the type commonly associated with good thickener practice cannot be made. True thickening requires a *line settling* of the pulp bed to expose a clear supernatant. All small particles are collected from the supernatant by the dense descending blanket of flocculent slimes, and large particles are sustained by the yield strength of the blanket. The net result is that all particle sizes settle together, with essentially no classification, in a thickener after the initial sedimentation of the coarsest particles at the feed well. Even at the feed well this sedimentation is restricted to those particles that are heavy enough to overcome the yield strength of the pulp. In a cyclone, even the small particles possess enough centrifugal force to exceed the yield strength of the pulp, and mechanical dispersion eliminates the col-

Table III—Typical Results from Sydney Cyclones

Control Feed Pressure, 18 Psi—Cyclone, 48 In., 20°—Vortex Finder, 8 In.—Feed Area, 40 Sq In.—Apex, Needle-type

TEST	1	2	3	4	5	6	7	Avg
Feed								
Solids, pct	8.4	19.4	15.4	19.8	23.0	21.4	32.6	20.0
Mesh, pct								
+20	0.0	1.6	0.6	5.2	1.8	4.3	3.8	2.5
+28	1.3	3.4	1.9	9.0	3.9	8.1	8.7	5.2
+35	2.6	8.3	5.9	16.8	11.7	16.9	18.5	11.5
+48	4.8	17.0	15.3	28.6	25.6	29.7	32.9	22.0
+65	12.6	31.1	32.3	42.8	41.1	46.3	51.5	36.8
+100	24.2	53.3	50.5	66.8	66.6	70.2	76.0	58.3
+150	34.1	60.2	60.1	71.4	72.4	76.7	82.3	65.5
+200	37.5	63.4	64.1	73.4	75.2	79.3	85.4	68.3
−200	62.5	36.6	35.9	26.6	24.8	20.7	14.6	31.7
Overflow								
Solids, pct	7.1	7.2	6.1	6.4	7.3	5.7	5.7	6.5
Mesh, pct								
+35	0.0	0.0	0.1	0.1	0.6	0.1	0.4	0.2
+48	0.0	0.1	0.2	0.2	1.6	0.2	0.7	0.4
+65	0.0	0.2	0.5	1.0	3.8	0.3	1.6	1.1
+100	0.1	0.7	1.0	4.9	7.9	0.5	3.7	2.7
+150	0.2	1.2	1.3	6.1	9.5	0.8	4.7	3.4
+200	0.4	2.0	1.9	7.3	10.9	0.9	5.6	4.1
−200	99.6	98.0	98.1	92.7	89.1	99.1	94.4	95.9
Underflow								
Solids, pct	23.0	52.4	44.2	54.6	62.2		50.0	47.8
Mesh, pct								
+20	1.7	6.1	4.0	4.0	2.6	3.0	5.9	3.9
+28	4.1	11.9	7.8	8.3	5.9	6.3	11.6	8.6
+35	8.9	20.4	14.1	15.6	13.4	13.1	20.5	15.1
+48	17.9	31.9	26.0	28.4	28.7	25.5	32.0	27.2
+65	35.2	50.7	48.0	47.3	48.3	49.7	48.4	46.7
+100	63.6	77.5	74.1	81.5	81.9	83.0	78.7	77.4
+150	74.5	85.5	83.1	88.1	88.1	90.5	87.1	85.3
+200	80.5	90.2	88.7	92.0	91.3	94.8	92.0	89.8
−200	19.5	9.8	11.3	8.0	8.7	5.2	8.0	10.2
Weight, recovery	0.463	0.606	0.717	0.78	0.80	0.834	0.924	0.745
+200 recovery	99.2	99.2	99.2	97.7	97.2	99.8	99.5	98.9
−200 elimination	85.6	81.5	77.5	76.7	71.8	79.3	49.2	74.5
Corrected to 71 pct underflow solids								
−200 elimination		88.5	89.2	85.8	77.2		77.1	83.5
Weight, recovery		0.67	0.674	0.76	0.79		0.889	0.755

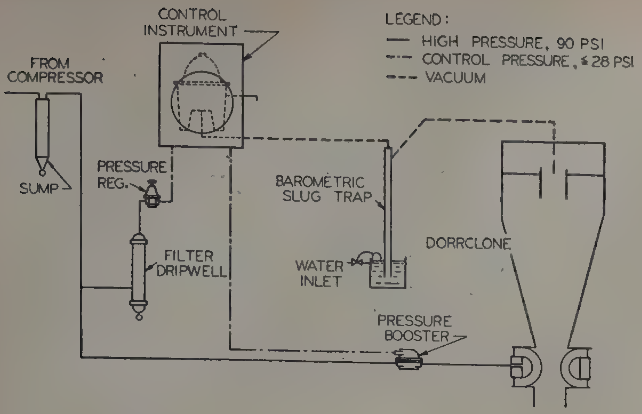

Fig. 5—Schematic of underflow density controller.

lecting together of slimes as flocs. Since the cyclone is always subject to classification instead of thickening, clear overflows cannot be produced with feed containing a natural full range of particle sizes. However, a clear overflow could be produced if the feed to the cyclone had been thoroughly deslimed in a previous operation so that there were no particles finer than the top size of the natural separation of the particular cyclone. Examples where overflows containing essentially no solids might result are: 1—Dewatering a concentrate in a cyclone operating so as to produce a top size of 20 microns if the flotation feed had been deslimed at 325 mesh or coarser. 2—Dewatering a pulp made up from a material, perhaps agricultural, which has a natural minimum grain size larger than the top size of the separation that would be made in the cyclone.

In the applications of the DorrClone discussed in this paper, data from field installations have been given, in some instances, while in others the data are from the laboratory.

Desliming

The DorrClone is especially applicable to desliming. The mechanical dispersion of the pulp eliminates settling of flocs (thickening) into the underflow. The dense underflows produced by the cyclone minimize the void volume in the underflow, thereby eliminating a maximum amount of the slimes that would be present in the pulp filling this voids volume. These two factors contribute to improved slime removals. The effect of centrifugal force to aid the small particles in exceeding the yield strength of the plastic pulp will allow high recovery of valuables without dilution of feed pulp. It was the combination of these factors that made possible the 91.7 pct slimes elimination and 97.5 pct recovery of valuables previously listed.

Where still more complete desliming is required, the DorrClone underflow may be repulped with water and fed to a second stage.

Florida Pebble Rock Phosphate

Details of American Cyanamid's Sydney units are given in a recent article by Crago.[1] The feed to the cyclones is washer debris at 14 to 17 pct solids and 35 to 45 pct —150 mesh. This is pumped from the surge tank at a rate of 9000 to 9500 gpm to a feed manifold. Pinch valves contained in 8-in. lines led to each of ten cyclones.

The feed entrance to the cyclone is 4 in. wide x 10 in. high. The vortex finder is 8 in. diam and the maximum apex opening is 5½ in. The overflow is removed to slime settling areas without further treatment. The underflow drops at 50 to 60 pct solids

Table IV. Typical Results from Homeland Cyclones

Control Feed Pressure, 14 Psi—Cyclone, 24 In. 20°—Vortex Finder, 6 In.—Feed Entrance, 20 Sq In.—Apex, Rubber Restricted

	1	2	3	4	5
Feed					
Solids, pct	14.2	22.1	25.5	20.2	14.2
Mesh, pct					
+20	1.7	2.1	3.5	2.2	2.3
+28	4.2	6.0	10.4	6.6	5.8
+35	8.4	15.7	17.6	15.1	11.4
+48	17.7	41.1	30.8	35.1	22.6
+65	33.7	64.4	48.3	56.9	39.9
+100	47.2	77.5	59.1	71.9	53.0
+150	54.5	80.6	65.4	75.5	60.2
+200	59.9	81.7	70.1	77.1	66.1
−200	40.1	18.3	29.9	22.9	33.9
Tons per hr (dry)	32.8	36.9	25.5	41.4	33.8
Overflow					
Solids, pct	6.5	5.8	4.4	5.1	5.5
Mesh, pct					
+35	0.0	0.73	0.3	0.35	0.18
+48	0.0	1.6	0.7	0.56	0.45
+65	0.26	4.8	1.1	0.98	0.89
+100	0.70	12.3	1.9	2.6	1.69
+150	1.6	17.6	4.0	5.0	3.11
+200	4.5	21.9	6.9	8.1	6.38
−200	95.5	78.1	93.1	91.9	93.62
Tons per hr (dry)	20.5	28.5	18.3	31.8	22.7
Underflow					
Solids, pct	70.2	69.8	71.7	76.6	72.8
Mesh, pct					
+20	6.0	3.8	7.1	4.8	8.2
+28	12.5	10.1	14.5	12.3	16.3
+35	21.3	23.0	24.5	24.5	26.4
+48	37.6	55.1	42.6	50.3	43.1
+65	60.7	81.4	67.0	76.2	65.6
+100	77.9	94.4	81.6	93.6	81.9
+150	86.9	97.4	89.3	97.2	89.5
+200	93.1	98.0	95.0	98.0	95.3
−200	6.9	2.0	4.97	2.0	4.7
Tons per hr (dry)	12.3	8.4	7.2	9.6	11.1
Weight, recovery	0.625	0.772	0.717	0.767	0.672
+200 recovery	97.1	93.9	97.2	97.5	96.8
+150 recovery	98.9	95.2	98.3	98.5	98.2
−200 removal	89.3	97.1	88.1	93.2	90.6
Feed, gal per min	794	721	765	776	742
Overflow, gal per min	723	670	702	684	675
Underflow, gal per min	71	51	63	88	67

Table V. Laboratory Tests for Desliming an Iron Ore

Test No.	1	2	3	4	5	6	7	Avg
Cyclone, in. diam	6	6	6	6	6	6	6	6
Feed entrance, sq in.	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
Vortex finder, in. diam	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Apex, in. diam	0.59	0.68	0.56	0.68	0.59	0.78	0.75	
Pressure, psi	35	35	35	35	35	35	35	35
Feed, pct solids	24	23.6	20.7	21.0	21.4	29.0	24.8	23.5
Overflow, pct solids	2.05	2.06	2.7	2.5	1.61	1.44	1.09	1.92
Underflow, pct solids	75	69.1	77.5	67.3	75.3	60.0	60.4	69.2
Weight underflow: weight feed	0.941	0.943	0.900	0.910	0.946	0.978	0.972	0.941
Feed, gal per min	56.6	55.6	49.5	49.1	54.5	50.0	47.3	51.8
Overflow, gal per min	47.1	45.1	43.1	41.1	46.6	33.2	34.3	41.5
Underflow, gal per min	9.5	10.5	6.4	8.0	7.9	16.8	13.0	10.3
Solids, sp gr	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2

Table VI. Results of Laboratory Tests for Desliming a Magnesite Ore

Test No.	1	2	3	4	5	6	7
Cyclone, in. diam	3	3	3	3	3	3	3
Feed entrance, sq in.	0.19	0.19	0.19	0.19	0.30	0.30	0.30
Vortex finder, in. diam	0.49	0.49	0.49	0.49	0.62	0.62	0.62
Apex, in. diam	7/16	7/16	7/16	7/16	3/8	3/8	3/8
Pressure, psi	10	20	40	60	15	30	60
Feed, pct solids	28.1	28.2	30.6	31.1	30.7	31.2	31.8
Overflow, pct solids	5.78	4.47	4.21	4.16	7.24	5.98	5.35
Underflow, pct solids	71.2	72.7	75.7	75.5	75.8	77.5	79.0
Weight underflow: weight feed	0.866	0.896	0.914	0.917	0.847	0.875	0.891
Feed, gal per min	10.1	11.7	15.2	19.3	15.9	22.0	30.4
Overflow, gal per min	7.8	9.1	11.6	14.7	12.6	17.3	23.7
Underflow, gal per min	2.3	2.6	3.6	4.6	3.3	4.7	6.7
Solids, sp gr	2.8	2.8	2.8	2.8	2.8	2.8	2.8

into a collection launder leading to a surge tank from which the sizer feed is withdrawn to a 12-ft quadruplex HX rake classifier.

The intelligence for the control of underflow density is the vacuum drawn at the top of the cyclone. This is relayed in this case through a Republic constant pressure regulator (Model P65S-AS 5714), which is integral with the cyclone. The control applies pressure at 35 psi to a piston that opens or closes the throttle valve into the apex of the cone. The total throw of this piston is 6 in., giving an opening of 0 to 100 pct of the apex area. This type of controller uses about 2 cfm (STP) of air per unit from a supply that may be 40 to 100 psi.

The DorrClone is subject to extreme wear at a point just above the apex valve. This wear is caused by the granular underflow pulp traveling at relatively high velocity. Of the several materials of construction used to withstand this wear, gum rubber has been by far the most satisfactory, as Table II shows.

A series of 24-hr tests were made on the Sydney installation. The results of these tests are reported in Table III. From the averages for these tests, it is seen that with a feed of 20 pct solids, 31.7 pct —200 mesh, 74.5 pct of the feed was recovered at 47.8 pct solids and containing 98.9 pct of the valuables and 25.5 pct of the slimes present in the feed. If the cyclone had been operated at 71 pct solids (common for the restricted aperture type valve), this slimes content would have been lowered to 16.5 pct of that present in the feed without affecting the recovery of valuables. Table III shows how the cyclone will accept feeds of widely varying composition and yield a consistent product and recovery. Each test has been recalculated for the underflow that would have been obtained with the standard DorrClone apex valve, the loss in weight recovery is purely at the expense of undesirable slimes, the valuables recovery would be unchanged.

Homeland Test

A 24-in. DorrClone unit was installed at the Homeland mine of the Virginia-Carolina Chemical Co. to deslime a —14 mesh phosphate rock ahead of flotation. The feed was taken directly from the washer distributor box, and there was therefore no chance for equalization of the per cent solids in the feed pulp ahead of the cyclone.

An 8x6 in. sand pump was driven at 1000 rpm by a 40-hp, 1750-rpm motor. The discharge of this pump was throttled by a 6-in. pinch valve, then fed

Table VII. Dewatering Mine Fill with a 24-In. DorrClone, Howe Sound Co.

Test No.	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Pressure, psi	34	33	32	40	39	38	37	40	38	37	30	29	28	27	35	34	33	32
Vortex finder, in. diam	2½	2½	2½	2½	2½	2½	2½	2½	2½	3	2½	2½	2½	2½	2½	2½	2½	2½
Feed area, sq in.	3.5	4.75	7.5	2.0	3.5	4.75	7.5	2	4.75	4.75	2.0	3.5	4.75	7.5	2.0	3.5	4.75	7.5
Feed																		
Solids, pct	42.3	44.2	44.8	46.6	43.1	45.6	47.8	44.5	44.7	45.5	44.3	46.1	44.7	45.5	45.1	45.8	45.5	44.7
Mesh, pct																		
+65	7.0	10.6	7.7	10.8	4.7	12.0	9.8	5.5	4.5	6.9	3.4	5.2	5.8	4.4	6.2	7.3	5.4	7.0
+100	21.1	25.1	22.0	25.6	16.2	26.8	25.3	19.5	17.1	21.7	17.6	20.4	21.2	18.2	20.7	22.3	20.3	23.1
+200	52.7	53.5	52.0	54.5	47.9	56.4	56.2	54.5	50.2	54.2	55.2	57.9	58.3	56.7	57.9	57.5	58.6	59.4
+325	69.6	68.8	67.4	69.4	60.4	71.0	71.4	70.2	67.1	70.4	72.4	73.1	72.9	72.8	73.3	73.2	73.2	73.7
—325	30.4	31.2	32.6	30.6	39.6	29.0	28.6	29.8	32.9	29.6	27.6	26.9	27.3	27.2	26.7	26.8	26.8	26.3
Tons per hr	41	45	53	39	43	47	55	39	47	47	33	38	42	50	36	41	45	53
Gal per min	286	299	341	238	286	295	324	254	303	301	216	234	270	315	229	256	284	342
Overflow																		
Solids, pct	20.5	17.3	22.3	22.0	18.1	14.7	19.7	13.9	14.7	19.6	21.0	20.8	14.5	13.2	18.6	16.6	16.3	12.9
Mesh, pct																		
+65	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.1	0.0	0.0
+100	0.7	0.3	0.8	0.5	0.2	0.1	0.4	0.2	0.2	0.5	0.7	1.1	0.3	1.1	2.0	0.7	0.7	0.2
+200	12.7	5.1	12.5	12.5	5.1	3.0	6.4	3.0	2.7	10.0	12.7	17.0	6.2	15.9	15.5	10.3	9.7	5.0
+325	30.6	17.3	29.2	30.8	18.2	11.8	17.8	11.5	10.6	28.8	31.4	34.2	18.1	33.1	30.2	27.7	23.1	15.0
—325	69.4	82.7	70.8	69.2	81.8	88.2	82.2	88.5	89.6	71.2	68.6	65.8	82.9	66.9	69.8	72.3	76.9	85.0
Tons per hr	30.2	36.6	39.7	30.3	34.4	40.5	47.4	34.0	40.2	38	25.6	30.4	36.4	41.0	33.1	34.7	38.4	47.0
Underflow																		
Solids, pct	67.5	69.1	67.9	69	68	68.8	69.0	66.4	66.7	66.6	65.4	66.7	65.9	66.6	66.3	67.3	65.3	66.1
Mesh, pct																		
+65	8.4	11.7	9.6	13.5	5.7	14.3	11.0	6.1	5.2	8.3	4.3	6.5	6.1	5.1	10.0	9.1	6.0	8.0
+100	25.7	26.2	28.4	32.2	20.2	31.3	28.4	23.2	19.7	26.2	19.3	24.3	22.4	21.9	28.4	26.4	23.3	26.0
+200	62.5	61.9	64.6	66.8	58.9	63.8	63.6	63.9	58.5	65.3	63.7	66.6	63.8	65.2	68.7	66.3	65.4	65.9
+325	78.6	78.2	79.7	81.3	77.9	79.4	79.3	80.0	77.1	81.9	80.0	81.4	79.8	81.4	83.3	84.0	80.5	80.3
—325	21.4	21.8	20.3	18.7	22.1	20.6	20.7	20.0	22.9	18.1	20.0	18.6	20.2	18.6	16.7	16.0	19.5	19.7
Tons per hr	10.8	8.4	13.3	8.7	8.6	6.5	7.6	5.0	6.8	9.0	7.4	7.6	5.6	9.0	2.9	6.3	6.6	5.0
Recovery	74.1	81.4	74.9	77.7	80.0	86.0	86.1	87	85.6	80.8	77.5	79.8	86.7	82.0	92.0	84.6	85.5	88.5
Separation, microns	132	107	130	126	103	91	114	93	90	123	131	142	111	142	154	129	128	104

through 6 ft of straight pipe to an adapter into which a pressure gage was tapped. The feed nozzle had a maximum opening of 8x3¾ in. Feed shims could be bolted into this nozzle to produce smaller open areas. Vortex finders of 5, 6, 8, and 10 in. diam were available; the 6-in. is the only one thoroughly tested to date. The overflow line was 10 in. at the spray chamber, reduced to 8 in. at the elbow. The 8-in. pipe ran off at 45° to a measuring box and then to disposal. The apex valve had a 4-in. maximum opening and was controlled automatically. The rate of discharge through this valve was measured by tilting a launder to collect the full flow for a timed interval in a weighing drum.

Over a period of three weeks five tests were made on this DorrClone by the Lakeland, Fla., laboratory of the Minerals Separation North American Corp. A summary of results is reported in Table IV. Tests 1, 3, and 5 were made with one dragline operating at three different positions in one mining area, Tests 2 and 4 with another dragline at two different positions in a mining area several miles away from the first. Therefore, in these tests, the characteristics of the feed were not only changing within any given test (the data of Table I were collected during Test No. 1 of Table IV), but the nature of the matrix was completely different from test to test. With the wide variation of average feed conditions 14.1 pct to 25.5 pct solids, 25.5 pct to 41.4 pct —200 mesh, the average underflow was not lower than 69.8 pct and was as high as 76.6 pct solids with a 97.5 pct recovery of valuables and 91.7 pct elimination of slimes. This was accomplished while rejecting the slimes at 4.4 pct to 6.5 pct solids. As a comparison, the hydroseparator at Homeland must not exceed 1.5 pct solids in the overflow or the losses of +150 mesh become excessive. Thus, the amount of water going to slimes settling when using a DorrClone is only 23 pct to 33 pct of that resulting from the hydroseparator.

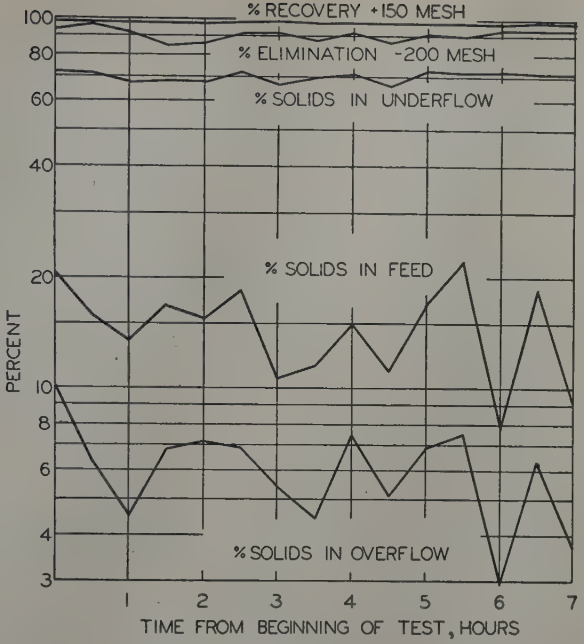

Fig. 6—Plot of Homeland data on desliming phosphate rock.

Iron Ore

Laboratory studies were made for desliming with a DorrClone ahead of flotation. A few tests, reported in Table V, were taken at random from one phase of this work. The results of flotation tests made on these products by the Mineral Dressing Laboratory of American Cyanamid showed good recoveries of iron, 89 to 94 pct, with a high grade concentrate, 59 to 62 pct Fe, 7.6 to 10 pct SiO_2.

From Table V, the 6-in. DorrClone, which had a 0.75 sq in. feed nozzle and a 1-in. vortex finder, operated at 35 psi feed pressure. It deslimed an

Table VIII. Typical Results at Plant X on Pyrite Concentrates

Test No.	1	2	3	4	5	6	7	9	10
Pressure, psi	11.5	11.5	11.0	11.0	11.0	12.0	8.0	15.5	19.25
Vortex finder, in. diam	1.375	1.375	1.375	1.375	1.375	1.375	1.375	1.05	1.05
Feed area, sq in.	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
Feed									
Solids, pct	20.2	23.8	14.5	6.14	14.5	22.78	18.0	20.7	20.1
Mesh, pct									
+48	0.4	1.6	0.0	0.4	0.4	0.4	0.4	0.4	0.4
+65	3.8	10.6	3.0	2.0	3.2	3.2	5.6	3.4	3.4
+100	17.2	25.0	11.5	7.2	10.6	16.8	20.2	16.4	15.8
+150	38.8	42.8	18.5	18.0	22.4	36.6	37.6	35.2	34.0
+200	61.2	57.0	42.5	30.0	42.2	51.6	59.6	50.0	49.2
−200	38.8	43.0	57.5	70.0	57.8	48.4	40.4	50.0	50.8
Tons per day (dry)	37.1	49.8	29.4	16.7	27.5	61.1	28.5	46.8	44.4
Pulp, gal per min	25.8	37.5	27.7	35.0	35.0	40.5	24.5	32.1	32.3
Overflow									
Solids, pct	2.8	5.0	3.71	2.29	3.37	5.33	2.90	3.98	3.03
Mesh, pct									
+48									
+65									
+100	0.0	0.4	0.0	0.3	2.0	0.30	1.4	0.2	0.13
+150	0.8	0.5	0.5	0.8	2.2	0.35	2.8	0.3	0.33
+200	1.6	1.7	1.0	0.9	5.2	2.10	3.5	1.35	0.73
+300	3.2	4.1	3.0	2.9	7.2	5.30	5.6	3.65	2.83
−300	96.8	95.9	97.0	97.1	92.8	94.70	94.4	96.35	97.17
Tons per day (dry)	3.9	7.5	6.4	6.5	5.2	10.7	3.6	6.85	5.37
Gal per min	22.8	33.4	25.5	33.8	32.5	35.7	21.4	28.2	28.6
Underflow									
Solids, pct	76.6	74.0	74.8	66.8	69.7	74.9	67.7	74.4	75.2
Mesh, pct									
+48	0.4	2.0	0.2	0.2	1.0	0.4	0.4	0.4	0.6
+65	4.4	12.2	4.2	2.0	4.4	4.0	5.4	3.8	4.4
+100	19.6	28.4	16.4	8.0	18.0	21.0	20.4	18.8	19.4
+150	36.2	41.2	32.0	22.4	39.4	44.6	41.2	41.2	40.6
+200	69.6	65.6	60.2	45.2	56.0	61.6	65.6	58.6	57.6
−200	30.4	34.4	39.8	54.8	44.0	38.4	34.4	41.4	42.4
Tons per day (dry)	33.2	42.3	23.0	10.2	22.1	50.4	24.9	39.95	39.0
Gal per min	3.0	4.13	2.2	1.25	2.47	4.81	2.96	3.87	3.7
Recovery, pct	89.4	84.9	78.2	61.2	80.7	82.5	87.4	85.4	87.9

Table IX. Typical Results at Plant Y on Zinc Tailings, 3-In. DorrClone

Test	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Feed pressure, psi	19	49	49	20	50	50	51	51	49	50	50	19.8	19.8	34.5	34.5	50.5
Vortex finder, in. diam	1.375	1.375	1.375	1.04	1.04	1.04	1.04	1.04	1.04	1.04	0.813	0.813	1.04	1.04	1.04	1.04
Feed area, sq in.	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.50	0.50	0.50	0.50	0.50
Feed																
Solids, pct	16.1	22.2	22.0	22.4	15.5	17.6	21.3	18.8	14.4	19.4	16.1	19.9	22.5	13.1	18.7	15.1
Mesh, pct																
+48	2.0	7.8	9.2	6.8	1.0	1.8	5.4	3.6	1.6	5.8	1.2	3.8	8.8	1.0	2.4	1.0
+65	7.6	17.8	9.2	18.2	5.6	7.6	16.4	12.0	4.8	14.8	5.8	14.2	21.2	5.2	9.6	5.6
+100	17.6	31.6	31.6	32.0	18.4	20.6	32.0	26.2	12.4	27.8	18.2	29.8	35.0	17.2	22.0	16.6
+150	29.8	41.2	40.2	42.4	29.6	33.0	44.0	37.4	22.0	38.0	30.0	41.0	44.4	29.8	33.2	28.6
+200	40.2	49.2	47.6	50.0	39.2	42.8	52.4	46.4	30.4	46.4	40.4	49.8	51.4	41.0	43.2	39.0
−200	59.8	50.8	52.4	50.0	60.8	57.2	47.6	53.6	69.6	53.6	59.6	50.2	48.6	59.0	56.8	61.0
Tons per day (dry)	46.6	129.9	130.9	60.7	49.8	67.3	92.8	80.8	56.1	78.2	46.5	34.5	52.5	37.9	65.8	42.0
Pulp, gal per min	45.0	82.9	85.0	39.5	55.5	59.7	62.5	63.0	58.9	58.7	45.6	25.5	34.3	40.0	43.4	48.1
Overflow																
Solids, pct	7.54	7.6	7.4	11.7	5.4	4.84	6.3	6.4	3.63	4.62	5.1	6.7	10.0	6.0	11.5	4.7
Mesh, pct																
+48	0.0			0.02	0.05	0.0					1.25	0.04	0.02	0.0	0.0	0.02
+65	0.1			0.08	0.35	0.03					1.60	0.08	0.06	0.03	0.03	0.04
+100	0.2			0.28	0.55	1.13					1.90	0.2	0.22	0.06	0.11	0.07
+150	0.4	2.5	2.0	1.28	0.85	1.38	0.4	1.0	1.0	1.0	2.15	0.5	0.42	0.14	0.35	0.37
+200	1.0			4.68	1.20	1.72					2.40	0.9	1.02	0.29	0.73	0.62
+300	3.6			13.48	2.00	2.33	1.2				2.80	1.3	6.42	0.64	1.74	1.37
−300	96.4	97.5	98.0	86.5	98.0	98.67	98.4	99.0	99.0	99.0	97.20	98.7	93.58	99.36	98.26	98.63
Tons per day (dry)	18.8	29.8	24.8	25.9	13.4	11.4	39.4	15.6	10.7	13.9	9.7	9.2	18.4	15.2	31.0	9.9
Gal per min	41.2	62.2	53.0	35.0	47.3	40.65	15.5	39.2	47.7	48.8	32.8	22.1	29.85	36.5	33.3	40.3
Underflow																
Solids, pct	68.3	50.5	38.8	69.7	50.2	37.2	44.3	38.1	40.8	54.6	36.6	69.6	69.6	65.7	41.9	47.3
Mesh, pct																
+48	3.2	10.2	6.0	11.4	1.2	2.4	6.4	4.4	5.4	6.4	1.4	4.4	13.8	1.2	3.0	1.2
+65	10.4	24.0	15.2	34.4	6.6	8.2	19.6	14.8	12.4	18.0	7.2	16.6	34.6	7.2	11.0	7.4
+100	28.0	38.0	27.0	58.0	20.6	26.6	36.8	31.8	25.8	36.4	23.8	36.6	56.2	23.4	26.4	24.2
+150	47.6	51.6	39.2	73.4	37.2	43.0	50.0	45.0	39.4	50.6	38.0	51.6	71.0	42.0	41.2	40.8
+200	63.8	61.4	50.0	83.8	52.2	56.0	59.8	56.2	51.4	61.8	50.6	63.6	81.8	58.6	52.6	53.2
−200	36.2	38.6	50.0	16.2	47.8	44.0	40.2	43.8	48.6	38.2	49.4	36.4	18.2	41.4	47.4	46.8
Tons per day (dry)	27.9	99.1	106	34.8	36.4	55.9	77.2	65.2	45.5	64.2	36.9	25.3	34.1	22.7	34.8	32.1
Gal per min	3.78	20.7	32.1	4.6	8.2	19.1	23.1	23.8	11.2	9.9	12.8	3.32	4.5	3.52	10.1	7.87
Weight recovery, pct	59.8	76.9	81.1	57.3	73.2	83.1	83.3	80.7	81.0	82.2	79.2	73.3	65.0	59.8	52.9	76.5

Table X. Fine Classification of Sand in the DorrClone

Material	**Sand**			**Mill Tailings**		
Test	1	2	3	4	5	6
Cyclone, in. diam	6	6	6	3	3	3
Feed nozzle, sq in.	0.75	0.75	0.75	0.19	0.19	0.75
Vortex finder, in. diam	1	1⅜	1⅜	0.49	0.49	1.38
Apex, in. diam	<½	<½	<½	7/16	7/16	¾
Pressure, psi	35	35	5	40	40	30
Feed, pct solids	2.6	2.16	4.42	32.6	23.9	25.4
Overflow, pct solids	1.6	1.6	4.0	16.3	9.21	13.2
Underflow, pct solids	71.2	55.0	49.4	70.3	69.0	71.6
Weight underflow: weight feed	0.391	0.271	0.103	0.65	0.71	0.591
Feed, gal per min	36.3	54.1	23.4	19.6	15.7	62.4
Overflow, gal per min	36.0	53.7	23.2	15.5	13.1	53.8
Underflow, gal per min	0.3	0.4	0.2	4.1	2.6	8.6
Separation, microns	25	32	42	35	27	74
Solids, sp gr	2.65	2.65	2.65	2.65	2.65	2.65

average of 51.8 gpm of 23.5 pct solids pulp, producing 41.5 gpm of reject slimes at 1.92 pct solids and 10.3 gpm of conditioner feed containing 94.1 pct of the feed solids at 68.2 pct solids.

Magnesite Ore

Laboratory tests were made for desliming magnesite ore using a 3-in. DorrClone. Seven tests from this study are reported in Table VI. The per cent solids in the feed and the nature of the feed are essentially the same in all tests. Table VI shows that for a given set of cyclone dimensions the throughput, the recovery, and the underflow density are functions of the feed pressure.

This application of the DorrClone removed all the objectionable fine slime with a perfect recovery of useful product of high density, 71 to 79 pct solids.

Dewatering

The DorrClone is convenient for dewatering or densifying where loss of the fines that will necessarily be contained in the overflow is permissible. A fluid pulp may be pumped to the site where the cyclone will extract the solids from the pulp and discharge them at a high, controlled density. The cyclone, which requires little more room or supporting structure than the pipe line itself, can be placed above conditioners, underground, or in small, inaccessible places where placement of conventional equipment would be impossible.

Mine Fill

At a Howe Sound property, test work is in progress to use the cyclone in dewatering a mine fill produced from a deslimed mill tails. Some results from a series of tests conducted by V. A. Zandon of Howe Sound are reported in Table VII.

A sand having good drainage properties has been produced by desliming the mill tails with a cyclone. This desliming would proceed on a 24-hr basis above ground where the sand would be stored. For 8 hr each day this sand would be washed at 45 to 50 pct solids into a sump, from which it would be pumped below ground. There, a cyclone located at the exact point where fill was to be placed would remove the solids from the pulp. The water containing some slimes could then be pumped away.

The tests reported in Table VII are concerned with the second or underground phase of this work. With a 24-in. DorrClone using 27 to 40 psi feed pressure and 42 to 48 pct solids in the feed pulp, Mr. Zandon has been able to discharge 33 to 53 tons per hr of coarse solids at 66 to 69 pct solids. As much as 92 pct of the solids pumped below ground were removed by the cyclone.

Pyrite Concentrate

A DorrClone has been installed as a test unit in Plant X to deslime and dewater a pyrite concentrate in preparation for further treatment. Results are given in Table VIII.

With a feed of 14.5 to 24 pct solids, 39 to 70 pct —200 mesh, a 3-in. cyclone operating at 11 to 19 psi will process 25 to 40 gpm of pulp. The underflow was

67 to 75 pct solids and 30 to 55 pct —200 mesh; and the overflow 28 to 5 pct solids, 92 to 97 pct —200 mesh. Under these conditions 61.2 to 89.4 pct of the feed reported to the underflow.

Zinc Tailing

The tails from a zinc flotation circuit are deslimed and dewatered in preparation for further treatment. Table IX contains results obtained from a 3-in. cyclone operated as a test unit in this plant. Pulp was processed at a rate of 50 to 130 tons per hr at 13 to 22 pct solids, 19 to 50 psi; 98 pct of the +200 mesh feed was recovered, and 50 pct of the slimes removed. An underflow of 68 to 70 pct solids was produced.

Fine Size Classification

The DorrClone is capable of producing separation in which the top size in the overflow is as fine as 10 microns. The top size in any specific instance is determined by the cyclone proportions and pulp densities but is easily varied and controlled. It can be used in almost any hydroseparation with the advantage that it produces cleaner and denser underflows than a hydroseparator and requires no chemical dispersing agent.

Sand

Tailor-made sands were produced by making fine size classifications on products from 1—a silica deposit and 2—a mill tailing of high grade quartz.

The data of Table X are presented as examples of the flexibility of the DorrClone resulting from changes in vortex finder, feed nozzle, pressure, and densities. Through adjustment of these factors a wide range of products and product qualities is possible.

The complexity of the relationship of the extremely large number of variables results in an apparent inconsistency from test to test with reference to capacity and separation. A complete understanding of this situation results only from an exhaustive study of each variable. Failure to do this has resulted in inexperienced investigators abandoning the cyclone in instances where it was later found to have great possibilities.

Silver Tailings

In an old silver tailings dump it was found that the coarse particles contained a core of high grade ore, but the fine particles had been completely depleted by the original leach and were worthless. It therefore was desirable to rework only the coarse particles. The DorrClone produced a fine size classification rejecting the fines. The recovery of the total silver varied inversely as the grade of the coarse product, of course, but was always considerably higher than the total weight recovery. These results are shown in Table XI.

Table XI. Fine Size Classification on Silver Tailings with the DorrClone

Test No.	1	2	3	4	5
Cyclone, in. diam	3	3	3	3	3
Feed nozzle, sq in.	0.30	0.30	0.30	0.75	0.75
Vortex finder, in. diam	0.62	0.62	0.62	1.38	1.38
Apex, in. diam	½	½	½	½	½
Pressure, psi	40	15	15	5	15
Solids, pct					
Feed solids, pct	19.7	19.3	18.2	29.6	30.3
Overflow solids, pct	7.0	7.0	8.6	23.3	25.4
Underflow solids pct	72.3	66.1	70.1	70.8	70.9
Weight underflow per weight feed	0.714	0.712	0.60	0.315	0.252
Feed, gal per min	21.3	11.3	14.2	25.9	46.3
Overflow, gal per min	18.6	9.8	12.8	23.5	42.9
Underflow, gal per min	2.7	1.5	1.4	2.4	3.4
Silver recovery, pct	78.0	76.8	68.4	46.4	40.3
Silver recovery: weight recovery	1.09	1.08	1.14	1.47	1.60
Silver per ton, oz.					
Heads	1.25	1.25	1.25	1.25	1.25
Overflow	0.93	1.01	0.90	0.92	0.90
Underflow	1.32	1.31	1.30	1.73	1.80
Solids, sp gr	2.65	2.65	2.65	2.65	2.65

Phosphate Rock

The DorrClone was used to produce a fairly good grade of brown phosphate rock from an old deposit left from an impounded debris. The particles within a marketable size range were of high bpl (bone phosphate of lime). Therefore, the cyclone was used to eliminate the finer sizes and recover the coarser sizes of particles. Approximately 90 pct of the solids were rejected, but about 25 pct of the phosphate was retained at a grade of about 60 bpl. The results are reported in Table XII.

Table XII. Fine Classification of Phosphate Rock with the DorrClone

Test	1	2	3	4
Cyclone, in. diam	3	3	3	3
Feed nozzle, sq in.	0.75	0.75	0.75	0.75
Vortex finder, in.	1.38	1.38	1.38	1.38
Apex, in. diam	⅜	⅜	⅜	⅜
Pressure, psi	5	10	20	40
Solids, pct				
Feed	7.10	7.22	7.00	7.43
Overflow	6.27	6.47	6.31	6.83
Underflow	46.9	48.7	49.7	50.4
Weight underflow: weight feed	0.135	0.121	0.114	0.095
Feed, gal per min	23.3	38.4	49.2	65.5
Overflow, gal per min	22.9	37.9	48.6	64.8
Underflow, gal per min	0.35	0.50	0.57	0.66
Phosphate recovery, pct	30.5	29.0	27.0	22.5
Phosphate recovery: weight recovery	2.26	2.40	2.37	2.37
Bone phosphate lime, pct				
Heads	27.2	26.2	25.8	24.8
Overflow	21.9	21.2	21.2	20.2
Underflow	61.5	62.8	61.2	58.6
Solids, sp gr	2.65	2.65	2.65	2.65

Degritting

Grits and cores may be removed effectively from a slurry or slip by the DorrClone. The increased acceleration resulting from centrifugal forces acting on the grits give them enough force to overcome the yield strength of the pulp. Grit removals of 90 pct or more of that entering with feed are common without excessive dilution of the pulp. Table XIII contains data collected from laboratory tests on this application.

Summary

With a DorrClone, flexibility of both mesh of separation, and volumetric capacity are possible while maintaining constant underflow density. This is possible through the use of an adjustable feed nozzle, interchangeable vortex finders, and a positive-acting automatic control of the apex aperture. This automatic control system uses as its intelligence a vacuum that exists as an inherent characteristic of the air core of an operating DorrClone and that can be correlated with underflow density. This control has been used with the tire type of adjustable aperture to hold the per cent solids of the underflow to within ±3 pct of its mean value when the feed per cent solids was fluctuating over a range of ±75 pct of its mean value.

Through automatic adjustments to yield maximum underflow density, the DorrClone squeezes out a maximum amount of the water present in the voids of its underflow and thereby eliminates a maximum amount of slimes from the underflow. This feature enables the DorrClone to eliminate in one stage as

Table XIII. Laboratory Degritting with the DorrClone

Material	Slaked Lime										Clay		
Cyclone, in. diam	6	6	6	6	6	6	6	6	6	6	3	3	3
Feed entrance, sq in.	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.375	0.375	0.375
Vortex finder, in. diam	1.38	1.38	1.38	1.38	1	1	1⅜	1⅜	1¼	1¼	0.5	0.5	0.5
Apex, in. diam	1	1	1	¼	⅞	13/16	1	1 1/16	1	1	0.25	0.25	0.25
Pressure, psi	60	30	30	35	35	20	20	10	25	15	35	75	75
Solids, pct													
Feed	12.6	14.9	14.6	8.03	37	38.2	34.0	34.7	34.4	33.2	29	29	30.0
Overflow	11.6	14.4	14.0	7.94	34.2	33.6	30.8	31.6	31.7	31.5	26.9	26.5	28.6
Underflow	23.1	19.5	16.9	51.74	63.2	61.1	64.1	58.3	65.7	56.6	38.2	40.8	41.4
Weight underflow: weight feed	0.154	0.115	0.232	0.014	0.165	0.267	0.179	0.195	0.152	0.115	0.248	0.252	0.155
Feed, gal per min.	71.2	47.3	37.6	64.2	40.6	34.7	43.4	28.7	40.4	34.3	8.86	13.9	13.5
Overflow, gal per min	65.6	43.3	30.3	64.1	37.1	31.7	40.0	26.4	37.4	31.6	7.31	11.5	12.1
Underflow, gal per min	5.6	4.0	7.3	0.1	3.5	3.0	3.4	2.3	3.0	2.7	1.55	2.4	1.4
Removal of refuse, pct													
+200-mesh					88.9	80.2	82.1	83.7	80.9	80.4	98.5	99.5	98.3
+325-mesh	96.9	95.2	92.3	85.6									
Size analysis													
Feed +200-mesh					0.81	1.90	1.90	1.50	1.91	2.04			
+325-mesh	4.78	3.06	2.79	0.26							3.01	2.95	2.32
Overflow +325-mesh	0.178	0.166	0.28	0.04							0.054	0.032	0.051
Underflow +325-mesh	30.7	25.3	11.14	15.9							11.94	11.72	14.7
Solids, sp gr	2.2	2.2	2.2		2.2	2.2	2.2	2.2	2.2	2.2	2.52	2.52	2.52

high as 90 pct of the slimes present in the feed pulp.

The mechanical dispersion and high centrifugal forces present in the DorrClone make it especially suitable for treating pulps that are naturally flocculent. This enables classifications as fine as 10 microns and degritting of plastic pulps with dilution at which the critical sizes in the feed would not settle in conventional equipment. Thus, without penalizing the effectiveness of the separation both overflow and underflow from the DorrClone can be of higher density than the products of a hydroseparator in the corresponding operation.

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But above all the author thanks E. J. Roberts whose understanding of the fundamentals of the cyclone formed the basis of this work and whose patience in consultation led to this writing.

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Correction

In the June 1951 issue: *Economic Aspects of Ground Water in Florida* by V. T. Stringfield and H. H. Cooper, Jr. P. 527, in the caption for Figure 1 credit should be given to David B. Ericson and the Florida Geological Survey for the preparation of the original map on which the figure is based.

Comparative Results With Galena and Ferrosilicon at Mascot

by D. B. Grove, R. B. Brackin and J. H. Polhemus

THE heavy media separation process plays an outstanding role in the concentration of 4000 tons of zinc ore per day at the Mascot mill of the American Zinc Co. of Tennessee. Of the total tonnage, 72 pct is treated in the heavy media separation plant to reject 56 pct of the ore as a coarse tailing, which has a ready market. Concentrates from this separation are beneficiated further by jigging and flotation. Approximately 25 pct of the total zinc concentrate production is made in the jig mill. Jig tailings are ground and pumped to the flotation circuit where the balance of the production is made. Fig. 1 shows a generalized flowsheet of the mill.

The Mascot ore is a lead-free, honey-colored sphalerite in dolomitic limestone, with lesser amounts of chert and some pyrite. A mineralogical analysis is given in Table I.

After 10 years of successful operation with galena medium and treatment of nearly 10,000,000 tons of ore, a decision to convert to ferrosilicon was made early in 1948 because of the increasing price of galena and consequent high operating costs. The conversion was made on Nov. 6, 1948, and the results obtained since that time have shown remarkable improvement over those made with galena. The comparisons given in this report cover the first 6 months of 1950 as representing the ferrosilicon operation, and the year 1947 as representing the galena operation. This was the last full year in which galena was used exclusively and is representative of the best work done during the 10 years of operation with this medium. After only 2 years' operating experience with ferrosilicon and treatment of 1,807,585 tons many advantages have been revealed and are summarized in Table II.

Table I. Mineralogical Analysis of Mill Feed, Pct

Calcium carbonate	49.5
Magnesium carbonate	35.2
Iron oxide and aluminum oxide	1.5
Zinc sulphide	4.5
Insoluble	9.3
	100.0

Table II. Comparative Data, Galena and Ferrosilicon

	Galena[a]	Ferrosilicon[b]	Difference
Operating costs per ton milled, ct.	21.21	9.12	12.09
Medium consumption per ton milled, lb	0.80[c]	0.15	0.65
Reagent consumption per ton milled, lb	0.45	0.02	0.43
Tailing assay, pct Zn	0.310	0.297	0.013
Concentrate, pct Zn	12.08	10.33	1.75
Heavy media separation recovery, pct	89.38	90.22	0.84
Mill feed rate, tons per hr	153	166	13
Heavy media separation feed rate, tons per hr	100	120	20
Tons milled per heavy media separation man shift	350	620	270
Mill feed to coarse tailings, pct	51.0	56.7	5.7
Lost mill time, pct	5.6	5.0	0.6
Power consumption, kw-hr per ton	2.06	1.92	0.14

[a] 1947.

[b] First 6 months of 1950.

[c] Net consumption after deducting credit for reclaimed waste galena. Consumption of new galena was 1.320 lb per ton milled. For entire life of galena operation, a credit of 40 pct of the value of the new galena added was realized from the sale of waste galena.

Development

Prior to the introduction of the heavy media process, all the mill feed was crushed through 5/8 in. and treated by jigging. A finished tailing assaying 0.66 pct Zn was made on rougher bull jigs, and cleaner jig tailings were ground for treatment by flotation.

The first test work on the sink-and-float method of mineral beneficiation was carried out at Mascot in 1935, using a 3-ft cone and galena medium for batch tests. The following year a 6-ft cone was installed for pilot-plant work. This unit became a part of the mill circuit on March 1, 1936, and handled a gradually increasing tonnage in the next 2 years as the process developed to the point where it could treat all the + ⅜-in. material in the mill feed. Coarse jigging was then discontinued on March 1, 1939, and all coarse tailings have been made by the heavy media separation plant since that time.

Feed Preparation: The original feed preparation plant consisted of a drag washer followed by two 4x10-ft Allis-Chalmers washing screens. A surge bin and two additional 5x12-ft AC washing screens were added in 1943. Use of primary and secondary washing screens was found essential to provide the cleanest possible feed for the cone and thereby avoid excessive contamination of the galena medium. Improved washing was obtained by replacing the drag washer with a 7x20-ft Allis-Chalmers scrubber, shown in Fig. 2, which has been in service since May 1944. Throughout the life of the galena operation, delivery of extremely muddy ore to the mill overloaded the medium cleaning system, and it frequently was necessary to cut off the feed and clean the medium for several hours until its normal viscosity had been re-established. The cleaning circuit

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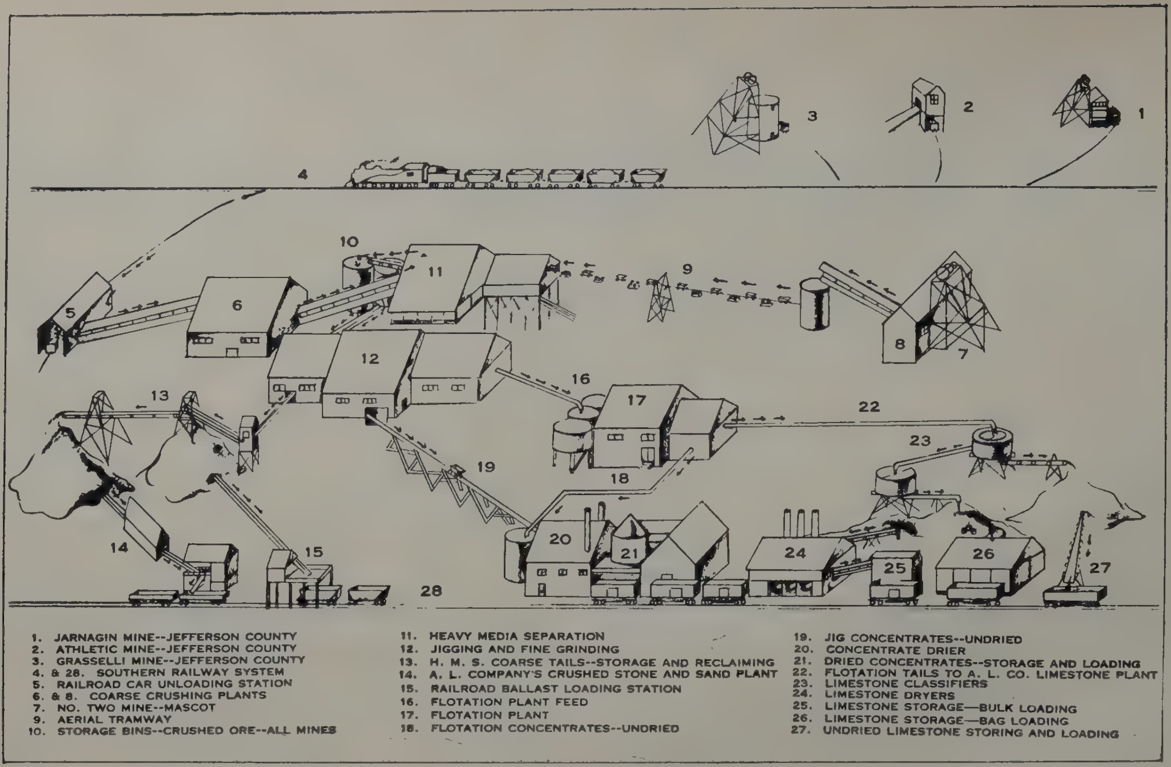
1. JARNAGIN MINE--JEFFERSON COUNTY
2. ATHLETIC MINE--JEFFERSON COUNTY
3. GRASSELLI MINE--JEFFERSON COUNTY
4. & 28. SOUTHERN RAILWAY SYSTEM
5. RAILROAD CAR UNLOADING STATION
6. & 8. COARSE CRUSHING PLANTS
7. NO. TWO MINE--MASCOT
9. AERIAL TRAMWAY
10. STORAGE BINS--CRUSHED ORE--ALL MINES
11. HEAVY MEDIA SEPARATION
12. JIGGING AND FINE GRINDING
13. H. M. S. COARSE TAILS--STORAGE AND RECLAIMING
14. A. L. COMPANY'S CRUSHED STONE AND SAND PLANT
15. RAILROAD BALLAST LOADING STATION
16. FLOTATION PLANT FEED
17. FLOTATION PLANT
18. FLOTATION CONCENTRATES--UNDRIED
19. JIG CONCENTRATES--UNDRIED
20. CONCENTRATE DRIER
21. DRIED CONCENTRATES--STORAGE AND LOADING
22. FLOTATION TAILS TO A. L. CO. LIMESTONE PLANT
23. LIMESTONE CLASSIFIERS
24. LIMESTONE DRYERS
25. LIMESTONE STORAGE—BULK LOADING
26. LIMESTONE STORAGE—BAG LOADING
27. UNDRIED LIMESTONE STORING AND LOADING

Fig. 1—Generalized flowsheet of American Zinc Co. and American Limestone Co. operations.

was operated each weekend to remove the contaminants that had built up in the medium during the week.

Medium Cleaning: The galena was cleaned originally by flotation, but this method was abandoned late in 1936 because it failed to reject enough of the contaminants and also because the reagents used to float the galena created froth and an unstable medium in the separatory cone. Various combinations of screens, settling tanks and sand and slime tables were tried in the next 2 years, and eventually a satisfactory cleaning circuit was developed. Trisodium phosphate was used as a dispersant for slimes and proved invaluable in maintaining the fluidity of the medium. Clean galena from the tables was returned to a storage tank ahead of the cone, and the table tailings were pumped to a storage pond. Periodic cleanups of these sands were made for recovery of their lead and zinc values. Extremely fine galena from the final settling tank was also pumped to a pond and reclaimed yearly for shipment to a lead smelter. Throughout the life of the galena operation, the returns from this reclaimed material amounted to 40 pct of the cost of the new galena used.

Conversion to Magnetic Medium

Following the decision to convert from galena to ferrosilicon, plans for the new circuit were prepared under the direction of Robert Ammon, the company's Chief Metallurgist, and R. H. Lowe of the American Cyanamid Co., who followed all details of the changeover and to whom much credit is due for the success of the new circuit. A 10½-ft cone was installed in July 1948 to replace the old 9-ft cone, see Fig. 3. All necessary equipment for the new medium cleaning circuit then was installed adjacent to the cone without any interruption of normal operations. The conversion to ferrosilicon took place on Nov. 6, 1948 over a regular weekend shutdown. The galena was pumped out of the circuit to storage ponds, and a thorough cleanup was made. This work was completed in three shifts. The circuit was filled with 60 tons of 100 mesh ferrosilicon on the following day, and a few necessary adjustments to the new equipment were made. The feed was put on at 2:30 pm on Nov. 8 and was not taken off again that day. The cone was handling its full tonnage rate within an hour after starting and no operating difficulties were encountered. The circuit as originally constituted has remained unchanged, except for the addition of a second magnetic separator. This machine was put in on a trial basis for the Stearns Magnetic Mfg. Co. Its performance compared favorably with that of the Dings machine, which was part of the original installation, and it has been kept in service as insurance against a breakdown on either of the separators. In addition, use of primary and secondary separators has been beneficial in keeping medium consumption at a very low level.

Ferrosilicon Circuit

The flowsheet of the circuit is extremely simple, see Fig. 4. The −2-in. ore from the mill bins is conveyed to a 7x20-ft Allis-Chalmers rotary scrubber, where it is subjected to intensive washing to liberate the fines from the coarser pieces of rock. The scrubber discharges onto two 5x12-ft Allis-Chalmers low-head washing screens equipped with 5/16-in. sq hole woven wire cloths. Oversize from these screens is conveyed to the cone. The tonnage rate as determined by a Merrick Weightometer is recorded on a Rateograph at the cone operating floor. The −5/16-in. material from the wash screens is dewatered by a 30 in. x 75-ft belt drag. The sands are delivered to the jig mill feed bin, and the −65 mesh drag overflow material is thickened and pumped directly to flotation.

The −2 +5/16-in. feed is treated in a 10½-ft separatory cone equipped with an 8-in. internal air lift for removal of concentrates. Details of the cone and air jet are shown in Figs. 5 and 6. Rakes rotating at 5 rpm are mounted on the air lift, which serves as the drive shaft. An air consumption of 90 cfm is required to elevate the concentrates to the top of the air lift. The air is supplied at 100 psi pressure by a 568-cfm Ingersoll Rand compressor driven by a 105-hp motor. This compressor was used for the galena operation, which required much higher air consumption. Although too large for the present operation, it has been kept in service rather than purchase a smaller unit. An adjustable air jet permits the operator to control the exact point at which air enters the air lift. This feature is valuable in controlling the gravity of the bottom medium, and thus the differential in the cone. At present the tip of the jet is ½ in. above the bottom of the 8-in. air lift, and excellent control is obtained at this point. The jet is made of 1½-in. stainless steel shafting with a ⅜-in. diam hole in the center to carry the air. The outer surface of the jet is threaded so it can be raised or lowered through a packing gland by turning a handwheel. Plows are mounted on the bottom of the rakes to prevent build-up of medium and tramp steel on the bottom plate. These plows are cut out of discarded ball mill liners and bolted to the bottom of the rakes. They normally last from 1 to 2 years.

The concentrate airlift discharges onto a 3x6-ft drainage screen, and the medium drains back onto the surface of the cone through a punch plate with 2-mm round holes. Use of the smallest possible hole at this point has proved advantageous in removing fine zinc from the bottom medium and preventing an accumulation of this material in the cone, with a resultant lowering of the bottom gravity. Washing of the concentrates takes place on a 4x10-ft screen using a ⅛-in. round hole punch plate. Washed concentrates are crushed through ⅜ in. x 3-ft short head Symons cone crusher operating in closed circuit with a 4x10-ft screen. The undersize from this screen joins the dewatered —5/16-in. material from the feed preparation screens and is conveyed to the jig mill.

Tailings overflow the cone through a weir 42 in. wide onto a 4x8-ft drainage screen equipped with a ¼-in. round hole punch plate. The depth of medium and rock in the weir is 4 in. during normal operations. The oversize of the screen passes to a 5x12-ft washing screen using ¼-in. sq hole cloths. The clean tailings are then conveyed to storage piles for sale as railway ballast and concrete aggregate.

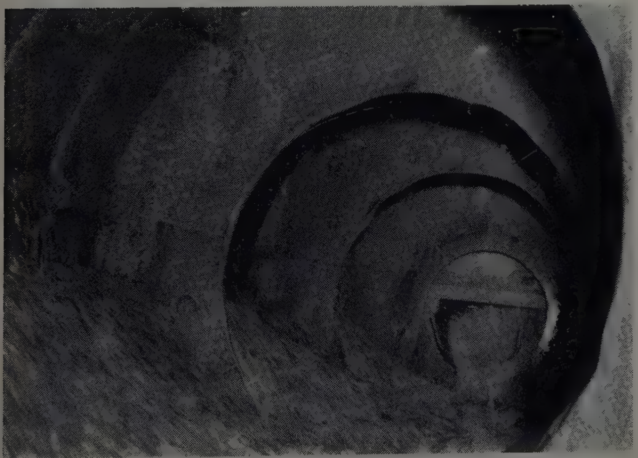

Fig. 2—Interior of 7x20-ft Allis-Chalmers rotary scrubber, looking from discharge towards feed end.

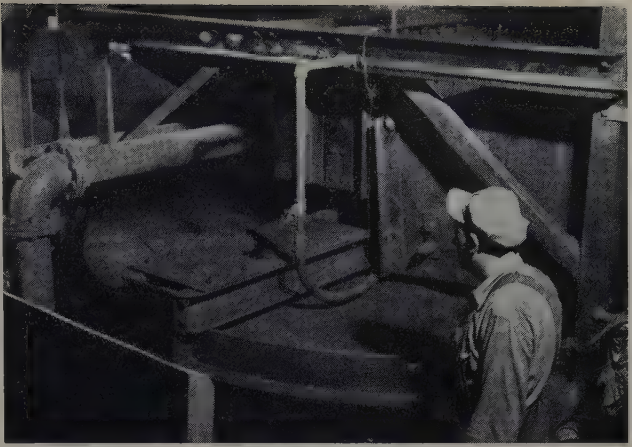

Fig. 3—10½-ft diameter separatory cone. Tailing overflow weir at left center, concentrate drainage screen at right.

The undersize from the tailings drainage screen enters a pump sump and is returned to the cone by a 6-in. Wilfley pump. A gate under the screen is used to divert a portion of the undersize to the cleaning circuit, and in addition, a continuous light overflow from the pump sump goes to the cleaning circuit. All the undersize from the tailing and concentrate wash screens is handled by a 4-in. Wilfley pump, which delivers this contaminated medium to a 20-ft Dorr thickener. Before entering the tank, the material passes through a permanent Alnico magnet, which causes the ferrosilicon particles to flocculate and settle rapidly. The thickened, dirty medium is pumped by a 2-in. Wilfley pump to the primary magnetic separator, a 24-in. Stearns machine. Its tailing is retreated in a 36-in. Dings Crockett magnetic separator shown in Fig. 7. Clean medium from both machines enters a 48-in. Colorado Iron Works densifier, which thickens it to 3.20 sp gr and delivers it through an ac demagnetizing coil to the medium pump sump for return to the cone. Overflow from the densifier is returned to the 20-ft Dorr thickener. The tailings and overflow from the secondary magnetic separator, which contain excellent values in fine zinc, are dewatered and handled in the jig and flotation circuits. Thickener overflow water is used on the concentrate and tailings wash screens.

Screen analysis and zinc distribution in the heavy media separation plant are given in Table III.

The five pumps in the circuit have the following functions: (1) 6-in. Wilfley, returns cone overflow medium and densifier discharge to cone. (2) 4-in. Wilfley, delivers dirty medium to 20-ft thickener. (3) 2-in. Wilfley, delivers thickened dirty medium to magnetic separators. (4) 2½-in. Fairbanks-Morse, thickener overflow to sprays on wash screens. (5) 1½ RV-5 Cameron motorpump, clean water to sprays on magnetic separators.

A cleanup elevator with an 18 in. x 70-ft belt is used for three purposes: 1—To handle all cleanup material from the basement, 2—To elevate the undersize from the tailings wash screen for delivery to the 4-in. pump, and 3—To add new medium to the circuit. The elevator discharge passes through the concentrate wash screen for removal of all +⅛-

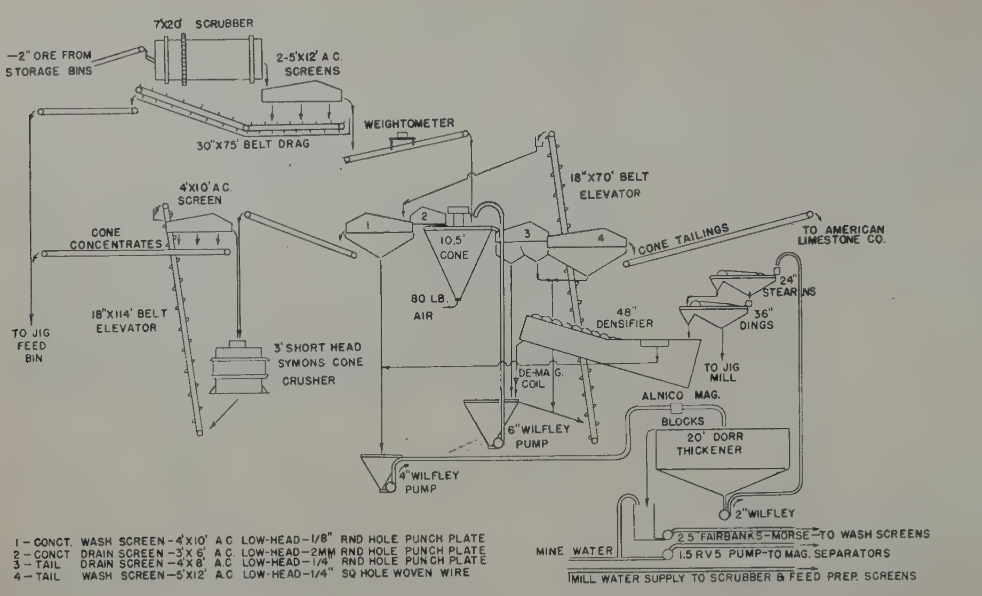
7'X20' SCRUBBER
—2" ORE FROM STORAGE BINS
2-5'X12' A C. SCREENS
30"X75' BELT DRAG
WEIGHTOMETER
4'X10' A.C. SCREEN
CONE CONCENTRATES
TO JIG FEED BIN
18"X114' BELT ELEVATOR
3' SHORT HEAD SYMONS CONE CRUSHER
1
2
3
4
10.5' CONE
80 LB. AIR
18"X70' BELT ELEVATOR
CONE TAILINGS
TO AMERICAN LIMESTONE CO.
24" STEARNS
36" DINGS
48" DENSIFIER
TO JIG MILL
DE-MAG. COIL
ALNICO MAG.
BLOCKS
6" WILFLEY PUMP
4" WILFLEY PUMP
20' DORR THICKENER
2" WILFLEY
2.5" FAIRBANKS-MORSE—TO WASH SCREENS
1.5 RV5 PUMP-TO MAG. SEPARATORS
MINE WATER
MILL WATER SUPPLY TO SCRUBBER & FEED PREP. SCREENS
1 - CONCT. WASH SCREEN - 4'X10' A C LOW-HEAD - 1/8" RND HOLE PUNCH PLATE
2 - CONCT DRAIN SCREEN - 3'X 6' A C. LOW-HEAD - 2MM RND HOLE PUNCH PLATE
3 - TAIL DRAIN SCREEN - 4'X 8' A.C LOW-HEAD - 1/4" RND HOLE PUNCH PLATE
4 - TAIL WASH SCREEN - 5'X12' A.C LOW-HEAD - 1/4" SQ HOLE WOVEN WIRE

Fig. 4—Mascot heavy media separation flowsheet.

in. material, and is then pumped to the medium cleaning circuit. A second elevator employing a 18 in. x 114-ft belt, raises the crushed cone concentrates to a 4x10-ft screen, which closes the circuit on this product. The housings of both elevators are made of 7 gage tunnel liner plate. This material, which is normally used for culverts, has proved ideal as an elevator enclosure. Each housing has elliptical ends and straight sides, with a major axis of 6 ft (parallel to sides) and a minor axis of 4 ft. The plates are 18 in. in depth and flanged for a bolted connection between rings. The elevator drive is mounted on a frame at the top of the housing.

The original circuit had only one magnetic separator, the 36-in. Dings machine. It gave excellent service and medium recovery of 99.9 pct. The 24-in. Stearns machine was located so it could be used as either a primary or secondary separator. It has proved more convenient to operate it as a primary separator, since no additional pumping is necessary. Its tailing is retreated in the Dings machine, which picks up any losses that may occur because of a temporary overload on the Stearns. Medium is being cleaned at a rate of approximately 9 tons per hr, with a loss of 13.2 lb per hr, or 0.95 g per gal of separator tailings. The exceptionally high recovery made by this circuit justifies the continuous use of two separators, although one is adequate to handle the load under normal circumstances.

Table III. Screen Analyses and Zinc Distribution, Heavy Media Separation Plant

Screen Size, In.	Wt, Pct	Cum Wt, Pct	Zn, Pct	Cum Zn, Pct	Total Zn, Pct	Total Cum Zn Pct,
CONE FEED						
1½	6.46	6.46	1.16	1.16	4.84	4.84
1	27.75	34.21	2.40	2.28	27.72	32.56
¾	24.12	58.33	2.21	2.23	22.22	54.78
½	21.36	79.69	2.46	2.31	21.93	76.71
⅜	10.32	90.01	2.70	2.35	11.63	88.34
5/16	4.80	94.81	2.52	2.36	5.04	93.38
¼	4.29	99.10	2.91	2.39	5.21	98.59
—¼	0.90	100.00	3.78	2.40	1.41	100.00
TOTAL	100.00		2.40		100.00	
CONE CONCENTRATES						
1½	4.46	4.46	7.98	7.98	3.70	3.70
1	27.01	31.47	9.05	8.90	25.42	29.12
¾	22.79	54.26	10.08	9.40	23.90	53.02
½	20.59	74.85	9.76	9.50	20.90	73.92
⅜	10.67	85.52	10.30	9.60	11.43	85.35
5/16	4.98	90.50	11.02	9.67	5.71	91.06
¼	5.12	95.62	9.80	9.68	5.22	96.28
—¼	4.38	100.00	8.15	9.62	3.72	100.00
TOTAL	100.00		9.62		100.00	
CONE TAILINGS						
1½	4.31	4.31	0.21	0.21	3.08	3.08
1	29.30	33.61	0.29	0.28	28.40	31.48
¾	23.74	57.35	0.25	0.27	20.50	51.98
½	21.05	78.40	0.33	0.28	23.38	75.36
⅜	10.62	89.02	0.35	0.29	12.44	87.80
5/16	4.84	93.86	0.31	0.29	5.12	92.92
¼	5.20	99.06	0.28	0.29	4.98	97.90
—¼	0.94	100.00	0.66	0.29	2.10	100.00
TOTAL	100.00		0.295		100.00	

The distribution of the total ferrosilicon loss of 0.15 lb per ton milled is shown in Table IV. Slightly more than 50 pct of the loss is in the tailing from the cleaning circuit. Medium adhering to the cone products represents 30.3 pct of the loss, and 16.5 pct is unaccounted for. The greater part of this material is probably lost through oxidation of the ferrosilicon.

All washing is done with Spraco ramp-bottom nonclogging nozzles, shown in Fig. 8, which have proved highly satisfactory in this type of service. No. 9R nozzles, each of which deliver 7.4 gpm at 10-lb pressure, are used on the feed preparation screens with 15 nozzles per screen in three rows of five nozzles per row. This gives a water consumption of 111 gal per screen, or 1.30 gal per ton of ore washed. No. 7R nozzles, each delivering 6.4 gpm at 18-lb pressure, are used on the concentrate and tailing wash screens. The concentrate screen has eight nozzles in two rows of four each, supplying a total of 51.2 gpm or 2.6 gal per ton of concentrates washed. There are 15 nozzles over the tailing wash screen in three rows of five each, delivering 96 gpm or 1.0 gal per ton of tailings washed. The total water consumption for all screen washing is 369.2 gpm. Water used in the scrubber amounts to 520 gpm.

Heavy Media Separation Circuit

The normal operating week in the heavy media separation plant begins at 3 pm Monday, after all necessary general mill repairs have been carried out on day shift. Operations are continuous until midnight on Friday or Saturday, depending upon whether the plant is on a 5 or 6-day week. After the feed is cut off, the medium is diluted and the air jet is opened wide to remove all rock from the cone through the air lift. Three extra bags of lime are added to the cone to prevent any possible hydrolysis and setting up of the medium while it is in storage. The medium circulating pump is then stopped, and all medium in the pump sump and cone is drained into the basement. These steps are completed in about 45 min. The medium is drained into the cleanup elevator and returned to the densifier through the regular cleaning circuit until the densifier is full. This procedure requires about 1 hr. The remaining medium, representing perhaps 25 pct of the total circuit load, remains in the basement over the weekend. Two hours are required on Monday to fill the cone from the densifier and wash up the medium in the basement, and the circuit is ready for operation at the end of that time. This method of shutting down the plant has proved very simple and satisfactory. The medium has been stored in this manner for as long as 2 weeks, and no difficulty was experienced in resuming operations. It is essential, however, to add an adequate amount of protective lime if the medium is to be inactive for any appreciable length of time.

Control of the cone operation is simple, and under all normal conditions consists entirely of adjusting the amount of dilution water being added to the surface of the medium in the cone. The densifier spiral is kept at the bottom of the tank, and excess medium is stored in the densifier only when the feed is off the cone and on weekends when the plant is down. The operator checks the gravities every hour, using a 1 liter density bucket and a scale graduated to give a direct specific gravity reading. The cone top gravity is kept at 2.83. A higher reading calls for increased dilution water, and a lower reading for a reduction in the water. Very little adjustment is necessary as long as a uniform feed to the cone is maintained. Normal practice is to clean about 9 tons of medium per hr, a quantity which is in excess of that actually required to keep the medium clean and maintain the desired viscosity. Thus a margin of safety is provided to take care of any increased fines or moisture in the cone feed. Each day 500 lb of new medium are added to the circuit, and if necessary more is used to maintain the required gravities in the cone. In addition, 50 lb of lime are used daily to prevent hydrolysis of the ferrosilicon. The new ferrosilicon and lime are mixed with water in a small tank equipped with a high-speed agitator. Preparation of the medium in this manner for approximately 1 hr before introducing it into the circuit insures complete wetting and conditioning of the fine ferrosilicon particles and prevents losses as the new medium passes through the cleaning circuit.

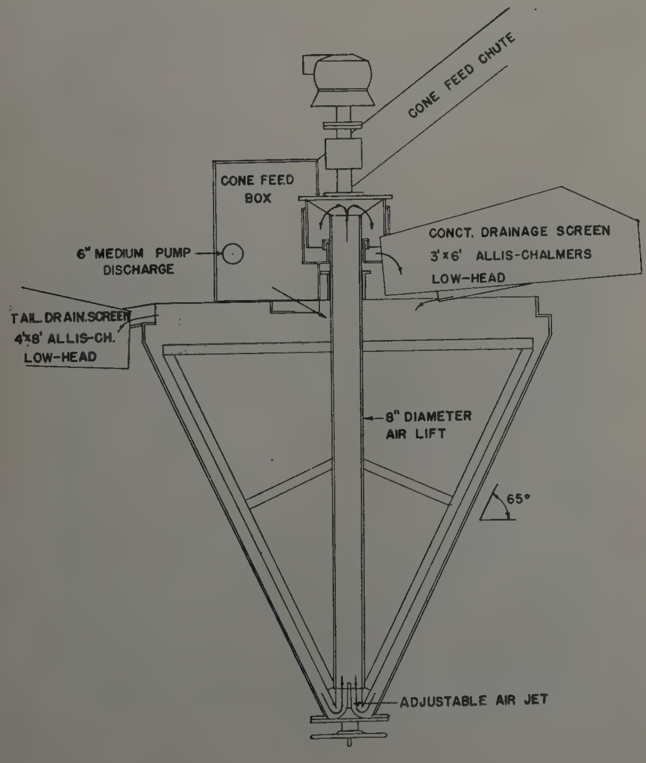
CONE FEED CHUTE
CONE FEED BOX
6" MEDIUM PUMP DISCHARGE
CONCT. DRAINAGE SCREEN 3'x6' ALLIS-CHALMERS LOW-HEAD
TAIL DRAIN.SCREEN 4'x8' ALLIS-CH. LOW-HEAD
8" DIAMETER AIR LIFT
65°
ADJUSTABLE AIR JET

Fig. 5—10½-ft diam separatory cone.

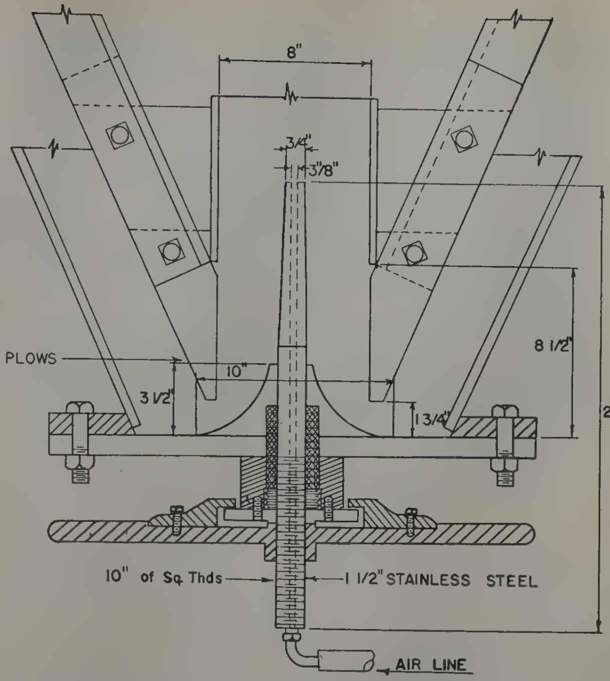

Fig. 6—Cone bottom showing relationship of air jet, air lift, and rakes.

When operations began, 60 tons of 100 mesh ferrosilicon were used to load the circuit. During the first few months the differential in the cone was appreciably higher than it is now, because of the newness of the medium and consequently greater average particle size. Approximately 10 tons of magnetite were added in the first 6 months to stabilize the medium and create a differential of about 0.10 between the gravities at the top and bottom of the

Table IV. Distribution of Ferrosilicon Losses

Product	FeSi, Lb		Total Loss, Pct
	Per Hr	Per Ton Milled	
Magnetic separator tailings	13.2	0.080	53.2
Cone concentrates	2.5	0.015	10.1
Cone tailings	5.0	0.030	20.2
Unaccounted for	4.1	0.025	16.5
TOTAL	24.8	0.150	100.0

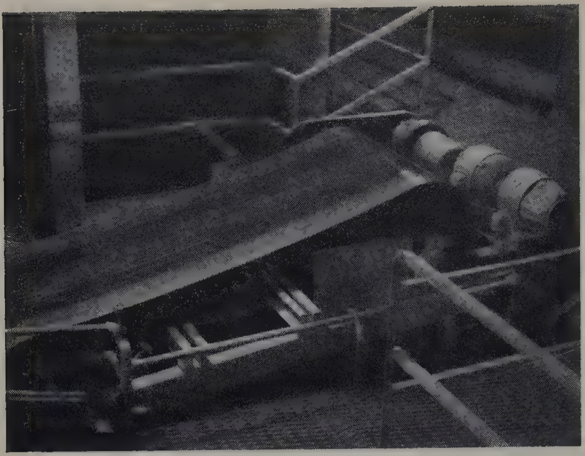

Fig. 7—36-in. Dings Crockett magnetic separator.

cone. After the medium had been in use for several months, its degree of fineness increased to a point where the condition of the early operating months was reversed and the differential began to drop. Changes were made in the circuit to permit cleaning of more medium, and in May 1949, 65 mesh ferrosilicon was substituted for the 100 mesh ferrosilicon formerly used for makeup medium. The coarser grade proved of value in maintaining the desired differential for several months. However, with medium consumption so low it has been difficult to maintain a differential of 0.10 during the past year.

The size of the medium in use in January 1950 is shown in Table V, which gives the results of sedimentation screen analyses run on the medium pump discharge, densifier discharge, and new 65 mesh and 100 mesh ferrosilicon. Pump data are given in Table VI. Only 3.8 pct of the pump discharge was retained on 200 mesh, as compared with 38.3 pct + 200 mesh in the new 65 mesh ferrosilicon. Breaking down of the coarser sizes is caused by attrition in the cone and pumps and usually is compensated for by normal addition of new medium. The rate of addition, which is governed by the medium loss, determines the average particle size in the circuit. At Mascot the medium consumption has been too low to maintain the average particle size that prevailed during the early months of the operation. However, the medium eventually stabilized after a year's operation at a size sufficient to maintain a differential of 0.09 between the top and bottom gravities. Recovery has been excellent and the operation very uniform at this level. A slight increase in differential would be beneficial to the concentrate grade but might have an adverse effect on recovery. This possibility will be investigated during 1951. However, recovery is of prime importance in the plant, and production of a high-grade concentrate is secondary to obtaining the lowest possible tailing assay.

Table V. Sedimentation Screen Analyses of Medium

Screen Size Intervals		New 65 Mesh Ferrosilicon		New 100 Mesh Ferrosilicon		Densifier Discharge		6-In. Medium Pump Discharge	
Microns	Mesh	Wt, Pct	Cum Wt, Pct	Wt, Pct	Cum Wt, Pct	Wt, Pct	Cum Wt, Pct	Wt, Pct	Cum Wt, Pct
200	+65	5.5	5.5	0	0	0.3	0.3	0.9	0.9
147	+100	11.1	16.6	0.6	0.6	0.3	0.6	0.4	1.3
104	+150	11.4	28.0	8.8	9.4	0.6	1.2	0.8	2.1
74	+200	10.3	38.3	21.0	30.4	1.6	2.8	1.7	3.8
44	+325	10.4	48.7	18.0	48.4	4.4	7.2	4.4	8.2
37	+400	26.1	74.8	16.3	64.7	14.2	21.4	20.8	29.0
27.8	+600	12.5	87.3	19.0	83.7	23.7	45.1	29.8	58.8
18.5	+800	8.3	95.6	12.9	96.6	45.8	90.9	36.0	94.8
16.2	+1000	2.0	97.6	2.2	98.8	4.9	95.8	3.4	98.2
8	−1000	2.4	100.0	1.2	100.0	4.2	100.0	1.8	100.0

Table VI. Pump Data

Pump	Connected Hp	Actual Hp	Rpm	Gpm
6-in. Wilfley	40	37	700	600
4-in. Wilfley	15	8.2	770	218
2-in. Wilfley	5	4.4	1160	150
2½-in. Fairbanks-Morse	20	14.7	1250	147
1½-in. RV-5 Cameron motorpump	5	3.7	3450	10

Data on heavy media plant screens are given in Table VII.

The cone overflow rate is 450 gpm of medium at a medium to rock ratio (by weight) of 3.4:1. The 6-in. medium return pump is handling approximately 600 gpm. The difference of 150 gpm between this and the cone overflow rate represents medium that is continuously bypassed back into the pump sump. The overflow rate can be changed if necessary by opening or closing the bypass valve. The concentrate air lift discharges at a rate of 215 gpm with a medium to rock ratio of 7.9:1.

Heavy Liquid Test Work

To determine the sink-float characteristics of the heavy media separation plant feed, a series of heavy liquid tests was carried out at Mascot in November 1949 by R. H. Lowe of the American Cyanamid Co. A composite sample of cone feed, fractionated in heavy liquid (acetylene tetrabromide) at specific gravity increments of 0.025 to 0.05, gave the results shown in Table VIII. Based on a specific gravity of 2.85, which is the true separating gravity for this ore, 84.32 pct of the feed was rejected as a product assaying 0.25 pct Zn. Under actual plant operating conditions, it is assumed that 1 pct near gravity sink material will report in the float. Since this near gravity material (2.85 x 2.90) in the composite assayed 3.50 pct Zn, it is calculated that the lowest average cone tailing that can be expected in the plant will assay 0.28 pct Zn. Under ideal operating conditions, this tailing can be made and has been

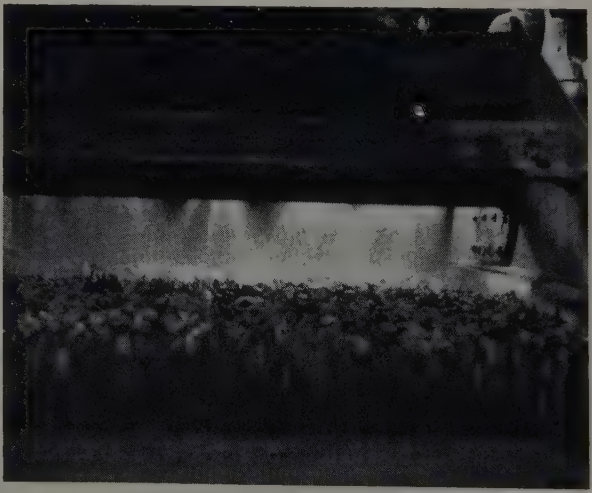

Fig. 8—Tailing wash screen showing action of Spraco ramp bottom nozzles.

Table VII. Heavy Media Plant Screens[a]

	Feed Preparation Screens (Two)	Concentrate Drainage Screen	Concentrate Wash Screen	Tailing Drainage Screen	Tailing Wash Screen
Size, ft	5x12	3x6	4x10	4x8	5x12
Tonnage per screen, tons per hr	85	20	20	100	100
Slope of screen, (up-feed to discharge), deg, min	2 30	4	2	2 30	2
Amplitude, in.	⅜	7/16	3/16	⅜	⅜
Sections per deck	3	1	1	1	3
Life in days					
Feed	26	13	22	16	27
Center	36				28
Discharge	26				29
Mechanism					
Size	3B	2B	2A	2C	3B
Hp	7½	5	5	5	7½
Type of cloth and size of opening	Woven wire, 5/16 in. sq holes, 0.135 in. diam wire	2 mm round hole, punch plate, 16 gage	⅛ in. round hole, punch plate 12 gage with ½ in. sq hole punch plate, 10 gage for protection	¼ in. round hole, punch plate, 10 gage	Woven wire, ¼ in. sq holes, 0.135 in. diam wire

[a] All screens Allis-Chalmers low-head, single deck.

Table VIII. Heavy Liquid Fractionation of Cone Feed, November 1949

Product	Wt, Lb	Wt, Pct	Cum Wt, Pct		Assay Zinc, Pct	Cum Zinc, Pct		Zn Recovered in Sink, Pct	
			Sink	Float		Sink	Float	Product	Cum
Sink 2.964	24.25	6.31	6.31		23.80	23.80		67.60	67.60
2.964x2.90	14.75	3.84	10.15	93.69	8.20	17.90	0.76	14.18	81.78
2.90x2.85	21.25	5.53	*15.68*	89.85	3.50	*12.83*	0.45	8.73	*90.51*
2.85x2.825	131.75	34.25	49.93	*84.32*	0.30	4.23	*0.25*	4.63	95.14
2.825x2.80	132.00	34.33	84.26	50.07	0.20	2.59	0.22	3.10	98.24
2.80x2.75	35.50	9.23	93.49	15.74	0.23	2.36	0.25	0.95	99.19
2.75x2.70	16.75	4.36	97.85	6.51	0.33	2.27	0.28	0.63	99.82
Float 2.70	8.25	2.15		2.15	0.18		0.18	0.18	0.18
TOTAL	384.50	100.00			2.22			100.00	

equalled or bettered in 3 months of the plant's 2-year life. The average tailing during this period has been 0.307 pct Zn.

The results of the above test have been plotted in Fig. 9, which gives the weight distribution and zinc recovery plotted against specific gravity. The curve on weight distribution shows that a drop in gravity from 2.85 to 2.84 can be tolerated, but below this point the weight of sink increases rapidly, and at 2.825 a fifty-fifty split by weight would result. This illustrates the necessity for extremely close gravity control in the cone.

A second series of heavy liquid tests was run in October 1950 for comparison with Mr. Lowe's original work. A composite sample of the cone feed was prepared, representing a week's operation in the plant. This sample was fractionated in acetylene tetrabromide at 2.85 sp gr. The sink-and-float portions then were screened and the sizes assayed. The results of this work are tabulated in Table IX.

Table IX. Heavy Liquid Fractionation of Cone Feed, October 1950

Size	Wt, Pct		Assay	Zn Distribution, Pct	
	Of Size	Of Total	Zn, Pct	Of Size	Of Total
2x1 in. sink	18.62	6.37	11.14	90.78	29.58
2x1 in. float	81.38	27.84	0.26	9.22	3.01
Total	100.00	34.21	2.29	100.00	32.59
1x½ in. sink	15.01	6.83	14.05	90.63	39.99
1x½ in. float	84.99	38.65	0.26	9.37	4.13
Total	100.00	45.48	2.33	100.00	44.12
½x5/16 in. sink	13.05	1.97	18.83	92.79	15.49
½x5/16 in. float	86.95	13.14	0.22	7.21	1.20
Total	100.00	15.11	2.65	100.00	16.69
—5/16 in. sink	12.66	0.66	22.18	93.07	6.09
—5/16 in. float	87.34	4.54	0.27	6.93	0.51
Total	100.00	5.20	3.04	100.00	6.60
Total sink	15.83	15.83	13.81	91.15	91.15
Total float	84.17	84.17	0.25	8.85	8.85
Total	100.00	100.00	2.40	100.00	100.00

The results on the cone feed sample check very closely with those obtained in the work done late in 1949, as shown in Table X.

Plant results in October 1950 show that the heavy media separation unit was actually making a tailing of 0.295 pct Zn and a concentrate of 9.62 pct Zn. Using the heavy liquid test on the cone feed as a basis, the highest theoretical recovery possible in the cone is 91.15 pct. Actual zinc recovery made in the plant was 90.49 pct or 99.28 pct of the theoretically recoverable zinc.

Advantage of Ferrosilicon Circuit

One of the chief advantages of the ferrosilicon circuit lies in its simplicity. When the conversion took place a major reduction in the number of pieces of equipment in service was made. Seven Allis-Chalmers low-head screens, two elevators, two surge bins, and a set of 54x20-in. Garfield rolls were removed from service permanently. The four

Table X. Heavy Liquid Fractionation of Cone Feed at 2.85

	Wt, Pct		Zn, Pct		Zn Distribution, Pct	
	Nov. 1949	Oct. 1950	Nov. 1949	Oct. 1950	Nov. 1949	Oct. 1950
Sink	15.68	15.83	12.83	13.81	90.51	91.15
Float	84.32	84.17	0.25	0.25	9.49	8.85

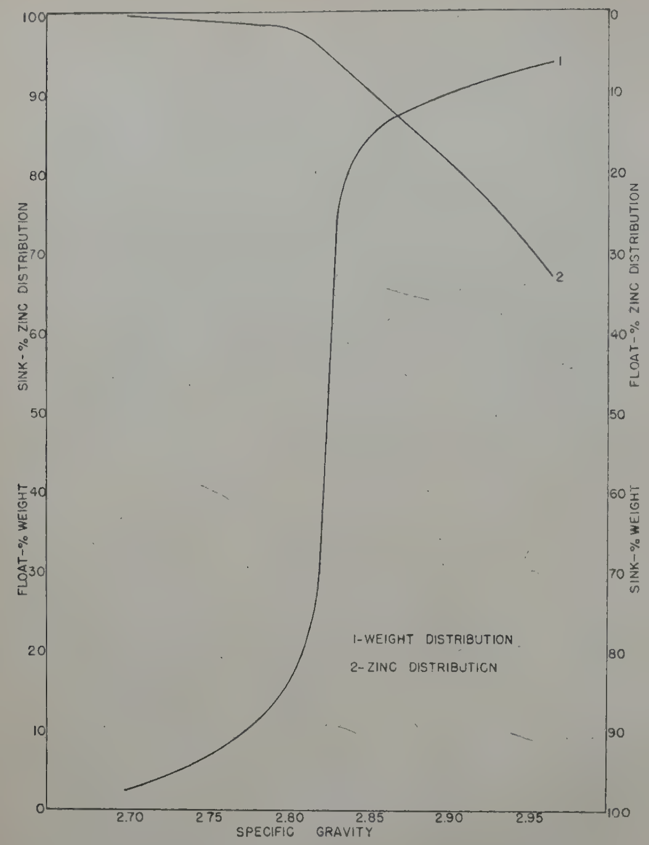

Fig. 9—Results of heavy liquid test on heavy media separation plant feed.

Wilfley tables and six decantation tanks in the old cleaning circuit were replaced with the densifier and two magnetic separators. The total number of pumps in use was reduced from 11 to 5, and the length of the conveying system in the heavy media separation plant and following it was reduced from 760 ft to 513 ft. A 200-ton steel surge bin in the jig mill with one Syntron feeder took the place of a 1000-ton wooden fine ore bin with 14 reciprocating tray feeders.

The efficiency of the magnetic separators used to clean the ferrosilicon permits the handling of extremely wet, muddy ore with little difficulty. Some of the ore received during the rainy seasons in the last 2 years would have taxed the galena cleaning system far beyond its capacity and shut the circuit down. This condition has never developed with ferrosilicon, and the plant has handled normal tonnage and made good results during the periods when the ore was at its worst. At present the mill is treating ores from four of the company's own mines in the area and is also handling two ores on a custom basis.

Table XI. Comparative Heavy Media Separation Costs Per Ton Milled, ¢

Operating	Galena	Ferrosilicon
Medium[a]	9.04	0.90
Reagents[a]	2.25	0.13
Labor	3.53	2.94
Power	1.29	1.30
Sundry	0.20	0.06
Maintenance and Repairs		
Screens and screen cloths	1.70	0.80
All other	3.20	2.99
TOTAL	21.21	9.12

[a] Costs for medium and reagents calculated on basis of prices prevailing at Mascot on Nov. 10, 1950, which were as follows, in cents per lb.

Galena	— 11.30
Trisodium Phosphate	— 5.00
Ferrosilicon	— 6.00
Lime	— 0.65

Another important benefit derived from the adoption of ferrosilicon has been the elimination of the possibility of contaminating the zinc concentrates with lead. Throughout the life of the galena operation there were periodic upsets in the plant which permitted galena to get into the jig and flotation circuits. The resultant high-lead assays on the zinc concentrates caused trouble at the smelter where lead-free zinc oxide is produced from Mascot concentrates. Every effort was made to prevent any leakage of medium into the main mill circuit, and all tailings from the cleaning circuit and fine galena from the final decantation tanks were pumped continuously to settling ponds some distance from the mill. Using ferrosilicon, the magnetic separator tailings are laundered directly to the jig mill circuit for recovery of their zinc content. Periodic reclaiming of this material from a pond has thus been eliminated and over 90 pct of its zinc content is now recovered. This compares with a recovery of about 70 pct made when the sands from the galena cleaning circuit were reclaimed and treated by flotation to obtain a bulk lead-zinc concentrate.

Costs

The comparative costs with galena and ferrosilicon are given in Table XI. These show the ferrosilicon cost to be 12.09¢ per ton less than the cost would be with galena at this time. The largest reduction is in the cost of medium and reagents, which is 1.03¢ per ton as compared to 11.29¢ per ton with galena. The reduction in operating labor resulted from the curtailment of three men at the time of the conversion to ferrosilicon. Although there has been a decrease of 0.14 kw hr per ton in power consumption, or 6.8 pct, the cost of power has increased slightly since 1947 because of higher local distribution charges.

Summary

Conversion to ferrosilicon has reduced heavy media separation operating costs by 12.09¢ per ton, given an improved recovery and higher tonnage rate, increased the percentage of the mill feed rejected as a coarse tailing, and provided a relatively simple circuit that has turned out consistently good results under all feed conditions. Further improvements in metallurgy are anticipated now that the early experimental phases have passed and the operation has become established on a routine basis.

Acknowledgments

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Cyclone Thickeners, A Practical Solution for Closed Water Circuit Operation

by Victor Phillips and James P. Blair

Cyclone thickeners have emerged from the development stage and now can be accepted as useful tools for the recovery of fine coal from a preparation plant circulating water. As primary thickeners in a closed water circuit operation, their advantages over conventional thickening methods are twofold—the lower initial investment and reduced area requirements.

CLOSED water circuit operation at the coal preparation plant of the United States Steel Co., Clairton Works, has never been optional. Prohibited by law from disposing of slurry to the adjacent Monongahela River and prevented by lack of available space from utilizing settling ponds, it has been mandatory that the system operate on a closed water circuit or not operate at all.

Lyons[1] has outlined in detail the generally accepted concepts of closed water circuit operation the chief of which is that "no portion of either the operating water or of the fine solids removed from the operating water is allowed to drain or 'bleed' directly to a stream or natural water course." This is literally true at Clairton since there is no existing outlet whereby slurry could be bled to the river. Whenever it has been necessary to drain the system, a limited impounding area located at some distance from the washing plant is used. At such time as when the cleaning operations are to be resumed, the water is returned to the system for recirculation. As a result of these limiting circumstances, this Rheolaveur preparation plant, sealed discharge for coarse coal and free discharge for fine coal, has operated under a handicap.

Normally the operation is scheduled to be carried on for 40 hr without interruption with the subsequent 8-hr period devoted to service and repair of the washing equipment. Although it has been possible to adhere to this planned routine most of the time, it has been necessary on occasion to stop the 40-hr run because of an excessive build up of very fine solids in the circulating water. Since it has been established that there can be no plant bleed and consequently no addition of equivalent amounts of fresh water to dilute the system, stopping the washing of coal and taking time to clean out the heavy burden of solids are the operators' only alternatives.

Until recently, the water clarification equipment consisted of two 70-ft diam Dorr thickeners, which have been processing about 3000 to 4000 gpm of overflow from the fine coal boot and the fine coal settling tank. Installed with the cleaning plant in the early 1930's, these thickeners have given nearly 20 years of continuous service and have been relatively free of serious operating and maintenance difficulties. Concentrated slurry from these units was collected and pumped for further dewatering to a pair of Genter vacuum filters which had a combined filtering area of 1400 sq ft. Within the past few years a 40x60-in. solid bowl centrifugal filter was put into service to augment the original equipment. In both instances the effluents were returned to the clarified water sump for recirculation in the plant.

The settling areas of the thickeners and the capacities of the filters were adequate during the earlier days of operation. As the proportion of fine solids in the raw feed increased, as a result of mechanized mining methods, the load to the water clarification system grew. The situation became more critical until it was deemed necessary to provide additional solids recovery equipment to compensate for the increased load. Careful investigation of available equipment and a study of performance in the field indicated that cyclone thickeners in conjunction with a disk-type vacuum filter were a promising combination. In addition to the economical advantages of the proposed system, the relatively small space requirements for a cyclone thickener installation were considered attractive in view of the fact that available areas for conventional settling devices were so limited.

Although the original patents for cyclone type thickeners date back to 1891,[2] the principles of these patents only recently have been put in general use in the coal industry. Simplicity of construction and operation have been intriguing features to those who have investigated their potentialities. Basic construction for cyclone thickeners provides a tangential entry tube into a cylindrical section of appropriate diameter. Attached to the lower end of the cylindrical section is a conical portion tapering to a relatively small opening. At this apex the concentrated

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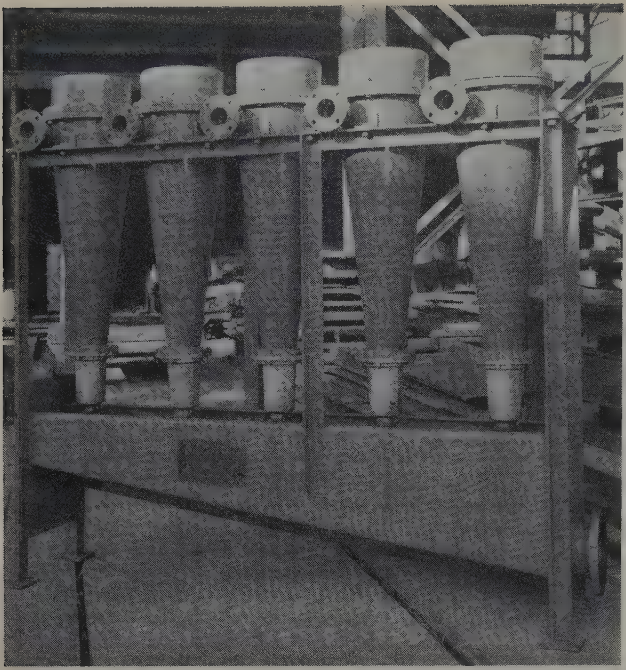

Fig. 1—Bank of five 14-in. diam cyclone thickeners. Capacity of unit is 1500 gpm.

solids are collected and discharged. At the top of the cyclone a cover plate with a downward projecting tube, generally described as a vortex finder or overflow orifice, provides an exit for the clarified water. When fed under pressure, high force fields are developed within the cyclone thickeners with the result that even very fine solids can be settled rapidly. The separated solids which tend to accumulate at the outer walls are then forced to the apex by means of the tapered section.

After a comprehensive engineering study of the entire circulating water system at the expected peak load, the decision was made to install five 14-in. diam cyclone thickeners, shown in Fig. 1, together with five units of 3-in. diam cyclone thickeners, each unit containing 22 individual cyclones, shown in Fig. 2. Both sizes are standard units manufactured by Heyl and Patterson, Inc., Pittsburgh. In conjunction with these cyclones, a 12-disk, 8 ft 6 in. diam Oliver American vacuum filter was installed to accommodate the recovered thickener products. The revised flow scheme is shown in Fig. 3.

By utilizing a 32x32 ft space formerly occupied by a 500-ton storage bin, the Clairton Works engineering department was able to house the entire cyclone thickener and vacuum filter system without requiring additional structures. Built on three levels the cyclones are situated on the uppermost floor, the vacuum filter is on the second, while the centrifugal slurry pumps and the dry vacuum pump are neatly grouped on the ground level.

Inasmuch as the cyclone thickeners were to operate in parallel with the Dorr thickeners, a portion of the Dorr feed was taken to the 14-in. cyclones. Thus by lightening the solids and water load to the stationary thickeners more effective use of that equipment could be realized. Since the take-off point for the 14-in. cyclone feed was at the bottom of the Dorr feed launder, it was expected that stratification would result in a feed that was more concentrated. This was confirmed during a test run when the feed concentrations were determined to be 15.6 pct and 11.1 pct for the 14-in. cyclone and the Dorr thickeners, respectively.

Mounted as an integral unit, the five 14-in. cyclones flowing about 1500 gpm at 40 psi feed pressure simultaneously thicken and classify the feed slurry. By gravity, the thickened underflow is directed to the American filter for further dewatering. On occasion the underflow has been so dense that cyclone feed has been used as push water in the collecting trough, which has a slope of 1½ in. per ft. This extremely heavy underflow is most pronounced when the underflow orifices of the cyclones are new and unworn, thereby presenting a minimum passage to the flow of fluid yet allowing the concentrated solids to flow freely. The clarified overflow of the 14-in. cyclones flows to a screened sump for pumping to the 3-in. cyclones where further recovery of solids takes place. A dismantled 3-in. cyclone is shown in Fig. 4.

Table I indicates the performance of the 14-in. cyclone thickener during a 40-hr period, Oct. 15 to 17, 1950, approximately 6 weeks after the additional water clarification equipment had been installed.

The construction of the large diameter cyclone thickener is as follows:

The outer shell is fabricated steel with the inner surfaces of the cyclone body lined with a ½ in. thickness of soft, abrasion-resistant rubber vulcanized into position. Abrasion resistance is extremely important since the destructive action of slurries moving at relatively high velocities is well known. Although the body itself is not completely free from abrasion, the principal wear occurs at the apex where the concentrated slurry emerges. Regarding the rubber lining of the body it can be stated that careful inspection of the cyclones after more than 2500 hr of actual service revealed an almost negligible amount of wear. This can be attributed largely to the fact that the top size of the solids passing through the cyclones has been about 16 mesh. At other installations where coarser sizes are being processed, the wear rate is reported to be somewhat higher.

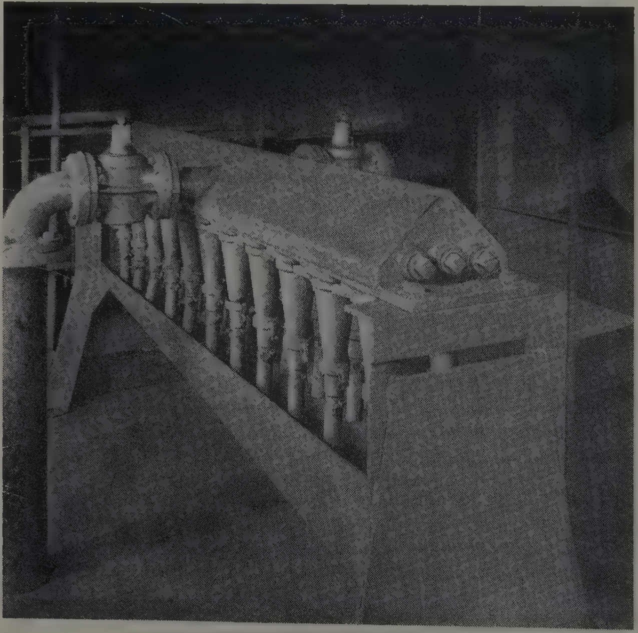

Fig. 2—Bank of 22 3-in. diam cones. Capacity of units is 250 gpm.

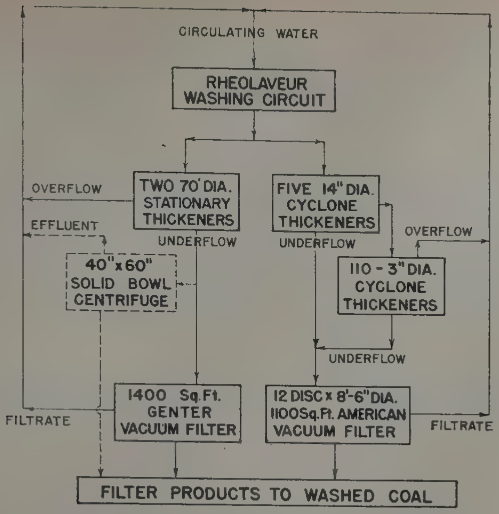
CIRCULATING WATER
RHEOLAVEUR WASHING CIRCUIT
OVERFLOW
TWO 70' DIA. STATIONARY THICKENERS
FIVE 14" DIA. CYCLONE THICKENERS
EFFLUENT
UNDERFLOW
UNDERFLOW
OVERFLOW
40"x60" SOLID BOWL CENTRIFUGE
110 - 3" DIA. CYCLONE THICKENERS
UNDERFLOW
FILTRATE
1400 Sq.Ft. GENTER VACUUM FILTER
12 DISC x 8'-6" DIA. 1100 Sq.Ft. AMERICAN VACUUM FILTER
FILTRATE
FILTER PRODUCTS TO WASHED COAL

Fig. 3—Water clarification system, Clairton Works coal preparation plant, United States Steel Co.

To compensate for the abrasion at the lower end of the cone, the underflow orifice has been designed as a replaceable part. Held in position by an aluminum retainer, the orifice can be replaced quickly after removal of two bolts. Originally the orifice piece was an all-molded rubber section but later developments indicated that it was a more economical arrangement to provide a small nihard orifice piece at the very apex of the cyclone. Thus the rubber orifice piece has become the mounting for the metal insert. Although the inserts have had to be replaced on an average of about 200 hr at this installation, the rubber mounting pieces appear to have a useful life well in excess of 1000 hr.

Continuing with the circuit, the purpose of the screen over the 3-in. cyclone thickener feed sump is to scalp any oversize foreign objects such as wood chips or other light gravity solids that conceivably could have been rejected as 14-in. cyclone overflow. Accumulation of material on the screen surface has not been a problem since the percentage of trash is unusually small. It has been demonstrated that solids that pass the ⅛-in. diam round hole openings of the screen surface will also pass through the openings of the 3-in. cyclone without danger of plugging. Although there was some difficulty with plugging during the initial operation of the system, in all cases the trouble was directly traceable to oversize solids, such as weld spatter and small portions of welding rods, which inadvertently had been permitted to get into the piping system during the erection. Now that the system has been purged of this material no further plugging problems have presented themselves.

Table I. Performance Data for 14-in. Diam Cyclone Thickener, Feed Pressure 40 psi

	Feed		Underflow		Overflow	
Size, Mesh	Wt, Pct	Cum, Pct	Wt, Pct	Cum, Pct	Wt, Pct	Cum, Pct
16x28	6.0	6.0	8.0	8.0		
28x48	15.0	21.0	30.0	38.0	Trace	
48x100	11.5	32.5	24.5	62.5	Trace	
100x200	7.0	39.5	16.5	79.0	4.0	4.0
200x325	5.5	45.0	7.5	86.5	8.0	12.0
325x0	55.0	100.0	13.5	100.0	88.0	100.0
Solids, pct	15.6		48.8		9.0	
Ash content, pct	13.7		10.6		17.2	

Table II. Performance Data for 3-in. Diam Cyclone Thickener, Feed Pressure 40 psi

	Feed		Underflow		Overflow	
Size, Mesh	Wt, Pct	Cum, Pct	Wt, Pct	Cum, Pct	Wt, Pct	Cum, Pct
100x200	4.0	4.0	15.0	15.0	2.0	2.0
200x325	8.0	12.0	22.0	37.0	1.0	3.0
325x0	88.0	100.0	63.0	100.0	97.0	100.0
Solids, pct	9.0		32.7		6.8	
Ash content, pct	17.2		11.9		19.2	

Table III. Vacuum Filter Performance Data

	American	Genter
Area, sq ft	1100	1400
Cake rate, tons per hr	30	38.7
Feed (solids, pct)	43	36.6
Cake moisture, surface, wet basis, pct	24.8	26.8
Filtrate (solids, pct)	Trace	2.8
Ash content, pct		
Feed	11.1	11.6
Cake	11.1	10.3
Filtrate		14.4

From Table II it is evident that the 3-in. cyclone feed consists principally of solids smaller than 200 mesh, which explains the use of the smaller, higher efficiency units. Driessen and Criner[3] have established that the collecting efficiency of cyclone thickeners is an inverse function of cone diameter, i.e., large diameter cyclones make a 50 pct separation at a larger particle size than will a smaller diameter cyclone, see Fig. 5.

Concentrates from the 3-in. cyclones are added to 14-in. underflows, and the combined solids are dewatered on the American filter to form a cake having a moisture content ranging from 19 pct to 24 pct with the average being about 21 pct. Filtrate from this filter contained only a trace of solids, which results from using a closely woven fabric as a filtering medium. This is in contrast to the effluent of the Genter filters, which was 2.8 pct. In this instance a relatively open wire cloth was used as a filtering medium. Table III indicates the relative performance of the two types of filters.

Although it might be complained that there is a multiplicity of 3-in. cyclones, there being a total of 110 arranged in groups of 22, compact arrangement and other unique design features make for quick service and easy maintainance. Individual cyclones are of die cast aluminum and are equipped with removable rubber liners as well as replaceable underflow orifices. Again, for reasons of abrasion resistance, rubber has been utilized for these parts. How-

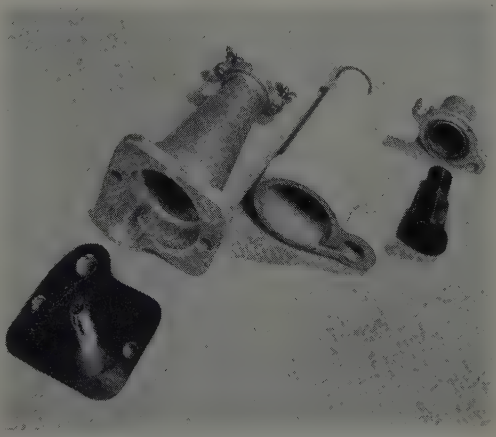

Fig. 4—Dismantled 3-in. diam cyclone.

ever, because the coarse solids have already been removed in the 14-in. cyclones, wearing of the 3-in. cyclone parts has been found to be very moderate. After more than 2500 hr of operation the original body liners are still in service and show only minor signs of abrasion. During this same period the rubber underflow orifices have been replaced twice, and the third set is still in service.

The clarified 3-in. cyclone thickener overflow is piped to the Dorr thickener overflow sump at which point the combined flows become the plant circulating water. At this point in the system the functions of the water clarification equipment end and their success or their failure becomes apparent.

Performance of this particular system has been such that the circulating water is maintained at substantially less than 10 pct solids concentration, whereas prior to the addition of the new facilities, 18 pct was considered to be an average condition. Furthermore, the reported figure of 18 pct represented a composite sample made up of small increments taken during the normal 40-hr operating period. The fluctuations that were once tolerated are now practically eliminated. Instead of a gradual build up of solids within the system, the combined facilities now remove the material at the rate at which it enters the circuit yet maintain a reasonably low level of circulating water concentration. Driessen and Criner[3] have given a complete explanation as to how it is possible to reach an equilibrium solids concentration in a system where there is recirculation of fine solids.

With this additional clarification equipment in service it has not been necessary to operate the horizontal type centrifugal filter, and it is hoped that at a later date only one of the two Genter vacuum filters will be required. Another operational improvement resulting from the installation of the cyclone system has been the reduction of shutting down time at the close of any operating period. A 2-hr clean-out period had been the practice in order that enough of the solids in the system would be removed so that the raking mechanism of the stationary thickeners could be stopped with the assurance that it could again be placed into motion. Since the residual backlog of slurry is now so much lower, only 15 min is required to purge the system.

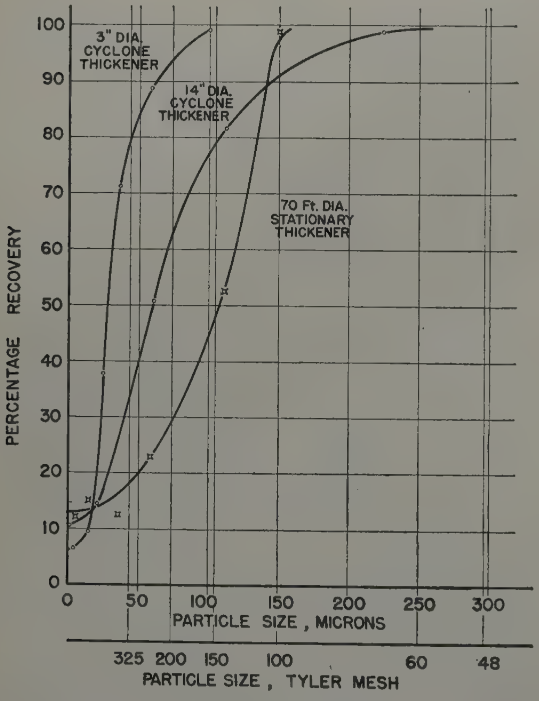

Fig. 5—Recovery characteristics, cyclone thickeners, and stationary thickeners.

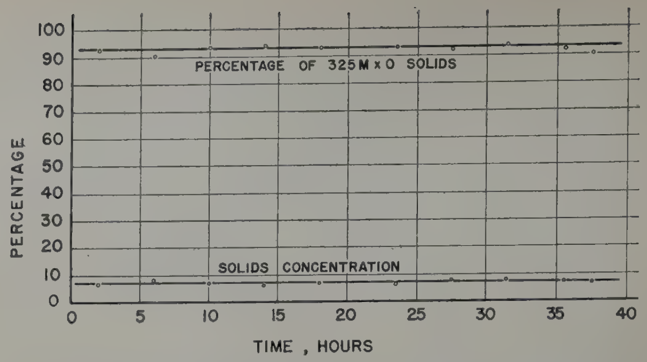

Fig. 6—Circulating water solids concentration and percentage of 325x0 solids during test period Oct. 15-17, 1950, Clairton Works coal preparation plant.

Uniformity of circulating water concentration is the hope of most preparation plant operators, but at Clairton it is a reality. Substantiating this statement are the analyses of samples of circulating water which were taken at frequent intervals during a 40-hr period. The results are shown in Fig. 6.

It is evident that the circulating water concentration remained within the narrow limits of 6 pct and 8 pct while the percentage of —325 mesh solids did not vary greatly from the 92.3 pct avg. That the percentage of 325 mesh x 0 solids as well as the solids concentration should remain nearly constant over a 40-hr period is most significant. It is factual evidence that the fine solids have not been building up without limit but rather have reached an equilibrium level, thus refuting the popular belief that recirculation of fine solids precludes the possibility of closing a water circuit.

Nearly 8 months of successful closed water circuit operation are now on the record as the result of this cyclone thickener-vacuum filter installation. Cyclone thickeners can no longer be considered laboratory novelties for they have emerged from this phase of their development and now must be accepted as a new and useful tool for fine coal recovery in the modern preparation plant.

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Acid Drainage from Coal Mines

by S. A. Braley

THE first commercial production of bituminous coal in the United States was in 1820, and formation of acid in the areas from which the coal was removed began at that time. Thus it is 130 years since the development of mine acid began. The coal operator has had to deal with the acid water produced in his mines, and his losses from corrosion of pumps and equipment have been tremendous. Other industries and the public have been forced to spend large amounts for their water requirements because of the acid discharge from the mines.

There are many contradictions in the literature on this subject probably because of variations in the analytical procedures followed and differences in the methods of calculation and reporting. The opinions previously expressed may be summarized as follows:

The production of mine acid has been looked upon as a simple chemical reaction wherein the oxygen of the air has reacted with pyrite to produce ferrous sulphate and free sulphuric acid. This reaction appears in many chemistry texts and generally has not been considered worthy of further study.

Since the measurement of hydrogen ion concentration or activity was developed and expressed in terms of pH, indicator and instrumental methods for pH determination have become readily available, making it natural to think of it as a measure of acid content.

Special purpose analytical methods applicable to potable waters, sewage effluents, and boiler waters, based in general upon the determination of what may be regarded as impurities, have been applied to more concentrated solutions constituting mine discharges, without reference to the fact that these specialized procedures had been developed to serve as operating guides for water-treatment plants.

It has been assumed that the pounds of acid delivered by a mine was roughly a constant varying with the number of acres mined, and that the concentration of the acid in the drainage was inversely proportional to the volume of flow. Increased flow was thought to result in a dilution of the acid produced.

Table I. Showing pH and Titratable Acid of Varying Concentrations of Ferrous and Ferric Sulphates

Ferric ion (Fe^{+++}) ppm	pH	Acidity ppm $CaCO_3$ equiv.	Ferrous ion (Fe^{++}) ppm	pH	Acidity ppm $CaCO_3$ equiv.
11,200	1.80	30,000	11,200	3.98	20,000
5,600	2.01	15,000	5,600	4.15	10,000
2,800	2.21	7,500	2,800	4.41	5,000
1,120	2.48	3,000	1,400	4.68	2,500
560	2.60	1,500	140	5.00	250
280	2.65	750	28	5.00	50
112	2.81	300			
56	3.07	150			
28	3.38	75			

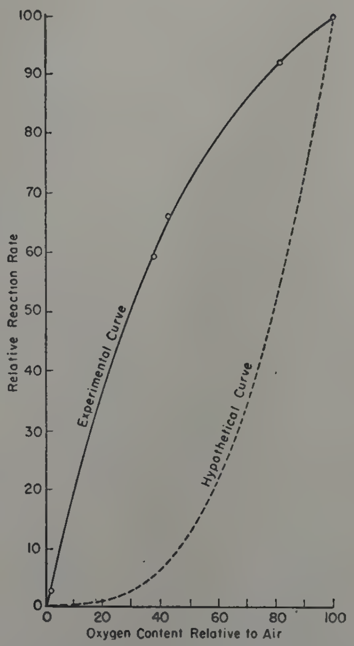

Fig. 1—Change in rate of acid formation with decrease in oxygen concentration.

When the Sanitary Water Board of the Commonwealth of Pennsylvania established a fellowship at Mellon Institute, the first approach to the problem was to review some of the chemical reactions involved and their relation to acid production. Since the sulphates of iron were among the principal constituents of coal-mine drainage, the first studies were based upon solutions of chemically pure ferrous and ferric sulphate. These are both neutral salts, that is, the iron and the sulphate are equivalent, but they produce acid reactions in water solution because as salts of strong acids with weak bases, they hydrolyze to form acid solutions. Solutions of each salt, of varying concentrations, were prepared and the pH, titrable acidity and iron and sulphate

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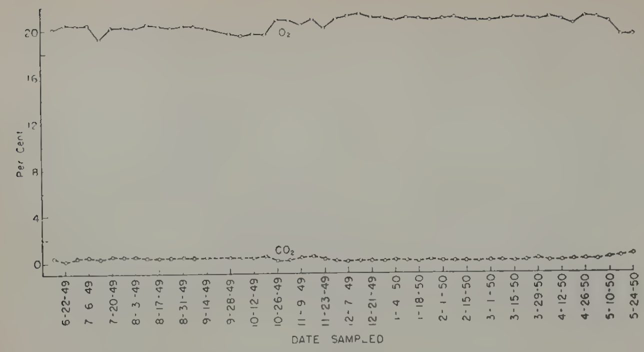

Fig. 2—Oxygen and carbon dioxide concentrations in Mine A for period sampled after sealing.

concentration determined for each solution. Titratable acidity was determined by titrating the hot solution to a phenolphthalein endpoint, using standardized sodium hydroxide solution, employing back titration with standard sulphuric acid when required. The data obtained are shown in Table I.

By definition, the pH of a solution represents the negative logarithm of the hydrogen ion concentration, the hydrogen ion concentration being expressed in terms of equivalents per liter. The hydrogen ion in these iron solutions is that derived from the ionization of the acid formed by hydrolysis of the iron salts, and both reactions are repressed by the buffer action of the unchanged salts present. During titration of such a solution with alkali, the pH (hydrogen ion concentration) remains practically unchanged until substantially all of the iron sulphates have entered into the reaction. Iron hydroxides precipitate and the liberated sulphuric acid is neutralized, causing the equilibrium to be maintained until neutralization is practically complete. As far as mine drainage is concerned, pH does not represent the titrable acidity, because the ferrous, ferric and aluminum sulphates, which are the principal acid-forming constituents of mine drainage, act as buffering salts.

A common source of sulphuric material in the coal mine is the so-called sulphur ball concretions found in the coal or in the banding rock. The sulphuritic material, which is found in the partings of the strata and disseminated through the associated measures as well as in the coal, is substantially of the same composition and is treated here as sulphur ball. Not all sulphur balls are actually sulphuritic. The sulphur balls that have been observed are about equally divided between those containing sulphuritic iron capable of producing sulphuric acid upon oxidation and those that are primarily calcium carbonate. It often is said that two mines on the same vein of coal separated by only a short distance produce different types of water, ranging from high acid on one side to high alkali on the other. This has also been found true in sections of the same mine.

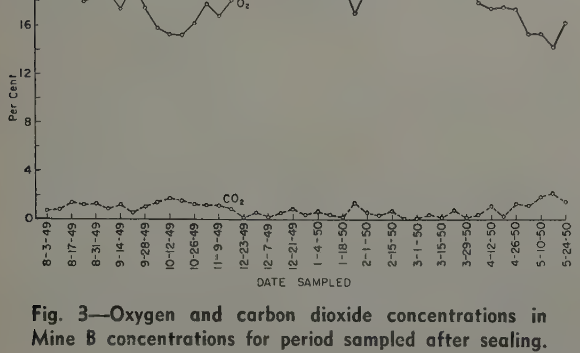

Fig. 3—Oxygen and carbon dioxide concentrations in Mine B concentrations for period sampled after sealing.

The points of entry of water into mines are difficult to find; in only a few cases could any exact point of entry be found. Whenever a point of entry has been found, the water coming from the undisturbed strata was alkaline rather than acid, a very good indication that the acid was produced within the confines of the mine. It was observed later that the amount of acid present in the water flowing or pumped from a mine represents that part of the acid production that has been permitted to come into contact with the transporting agency, the water, and is carried from the mine. When alkaline and acid water come from the same mine, a study of the geological strata and the condition of the section of the mines producing the different types of water usually will develop the answer to the question. If water entering the mine, like practically all ground water, is alkaline, that water, if it does not contact the coal face, the adjacent strata, or the products of their oxidation, will remain alkaline. Likewise, if no water enters the mine, no acid will be transported to the exterior regardless of the amount produced. Stream pollution by acid mine drainage is related directly to the water entering the mine, as the water must be contaminated before it can carry the contamination to the outside stream.

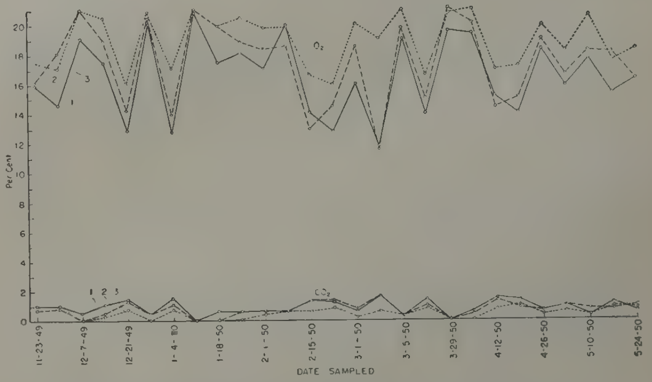

Fig. 4—Oxygen and carbon dioxide concentration of Mine C for period sampled after sealing. (1) 160 ft back of seal. (2) 120 ft back of seal. (3) 360 ft back of seal.

Study of a series of mines, with weekly sampling and gaging of their flow and complete analysis of each sample, indicated that the concentration of salts in the effluent from each mine was practically a constant for the mine, irrespective of season and flow, but the difference between two mines was often very great. This constancy of concentration for a given mine applies to ferrous and ferric iron, aluminum, calcium, magnesium, and sulphate. One point of extreme importance was the absence of free acid, that is, acid other than that produced by hydrolysis. This absence of free acid was determined by calculating the equivalents of the acidic constituents and equating them to the equivalents of basic constituents, the metals just referred to. From such an analysis the acidity of mine water may be determined in two ways, by calculating the equivalent of ferrous and ferric iron and aluminum,

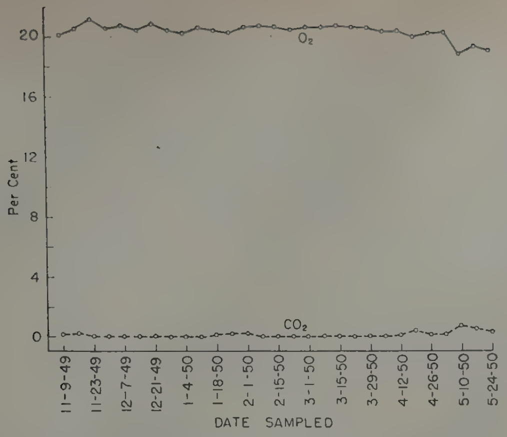

Fig. 5—Oxygen and carbon dioxide concentrations in Mine D for period sampled after sealing.

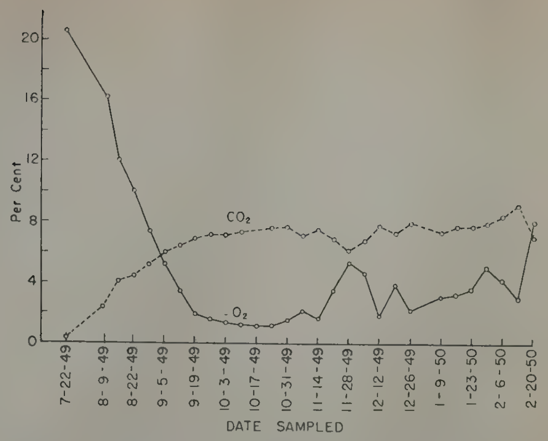

Fig. 6—Concentration of oxygen and carbon dioxide in Mine E for period sampled after sealing.

or by subtracting from the equivalents of sulphate the equivalents of calcium and magnesium. A few deep mines produce highly alkaline water with a chloride content as high as 1400 ppm. These waters are actually brines and fall outside of the scope of this discussion.

Theories have been advanced for the formation of acid as a result of bacterial activity. Those supporting this theory consider that some reaction other than normal chemical oxidation is necessary to explain the large quantities of acid produced. The results of Mellon bacteriological examinations are being reported elsewhere, but, thus far there is no evidence that bacterial activity plays an important role in acid formation. It is possible to credit chemical oxidation with all of the production of mine acid.

Rees and Kalinowski* reported that 10 to 12-in. cross-section samples representing the entire thickness of the coal seam of 14 different Illinois coals were stored in paraffin-coated boxes at 20 to 32°F for a period of 4½ years. During this time, under theoretically sealed conditions, the average increase in $SO_4^=$ content was 0.26 pct with a minimum of 0.05 pct and a maximum of 0.49 pct. This increase in $SO_4^=$ content may be considered as the equivalent of acid production, since it represents oxidation of the sulphur. This would represent a production of 316 lb of sulphuric acid a day, for the 4½-year period, from 100,000 tons of coal, stored in sealed containers. Thus, the acid produced by the oxidation of the sulphuritic material in pillar faces and gob remaining in a mine and openly exposed to air should be sufficient to account for all of the acid water draining from those mines.

The steps in the oxidation of sulphuritic material occurring in coal seams apparently are complex. Burke and Downs found that the reaction of dry air on dry sulphur ball material produced 1 mol of $FeSO_4$ and 1 mol of SO_2 for each mol of FeS_2 reacting. This has been confirmed, but it has been found further that in the presence of moisture the reaction produces 1 mol of $FeSO_4$ and 1 mol of H_2SO_4. Thus, one can assume that acid mine water should contain $FeSO_4$ and H_2SO_4. The water entering the mines has always been alkaline, with a definite concentration of alkaline salts of calcium, magnesium, and possibly sodium or potassium. The geologic strata in which the coal occurs contains shales and clays, together with certain amounts of calcite and limestone or dolomite, some capable of neutralizing the free acid, others being sulphuritic, all together producing a mine effluent consisting chiefly of the sulphates of ferrous iron, ferric iron, aluminum, calcium, and magnesium, and no free acid.

The chemical oxidation of sulphuritic materials is indicated by the equations

$$FeS_2 + O_2 > FeSO_2 + SO_2$$

or

$$FeS_2 + 3\tfrac{1}{2}\ O_2 + H_2O > FeSO_4 + H_2SO_4$$

It would appear that the rate of oxidation, or acid production, should be proportional to the oxygen concentration and, consequently, proportional to the partial pressure of the oxygen. There may be, however, several intermediate reactions that occur during the process so that their relative speeds may alter the apparent conditions. The Ohio River Pollution Survey of the U. S. Public Health Service, (1942), presents a hypothetical curve to show the change in rate of acid production with the change in oxygen concentration. Because this was presented as a hypothetical curve, experiments have been conducted to determine the relationship between oxygen concentration and acid formation. The results are shown in Fig. 1.

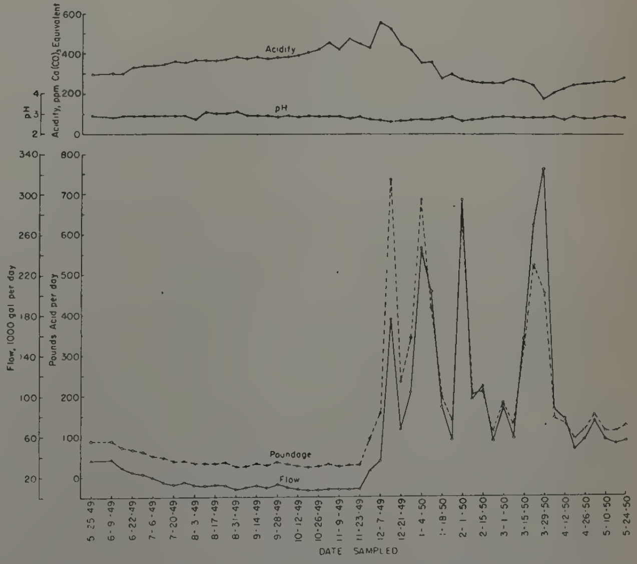

Fig. 7—Fluctuation in pH acidity flow and poundage of acid per day from Mine A.

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* Rees and Kalinowski: *Trans.* Ill. State Academy of Science, (1939) 32, No. 2, pp. 120-121.

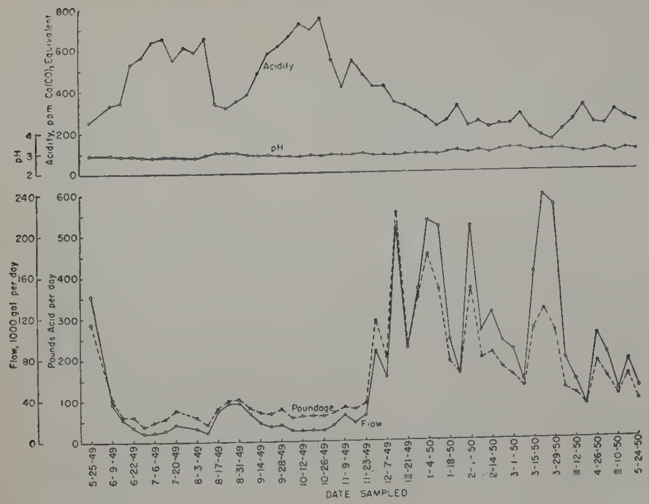

Fig. 8—Fluctuations in pH acidity flow and pounds of acid per day from Mine B.

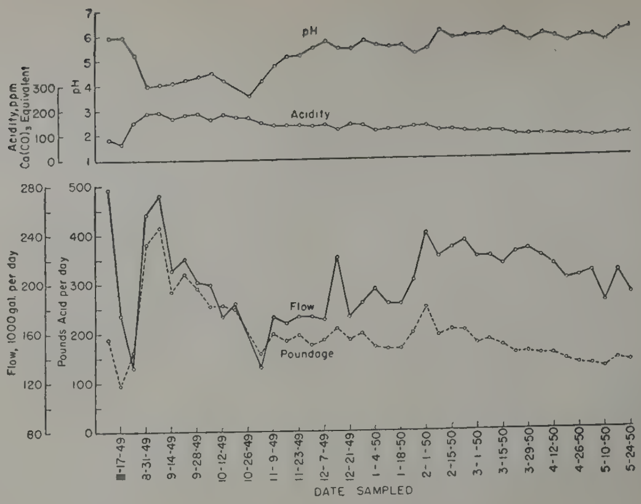

Fig. 9—Fluctuation in pH acidity flow and pounds of acid per day from Mine C.

This curve indicates the overall rate of reaction and acid formation with change of oxygen concentration. Pure air, 21 pct O_2 would produce 100 pct acid but zero oxygen, or air would produce zero acid, and a 15 pct oxygen concentration would be (12/21x100) 71 pct and produce 81.5 pct acid, or a reduction of 18.5 pct acid. Likewise 9 pct oxygen (9/21x100) 43 pct of 100 would produce 67 pct acid, a reduction of 33 pct. On the basis of the hypothetical curve, these reductions would be 40 pct and 92 pct, respectively. According to this experimental curve, exclusion of oxygen from a mine should reduce acid production. The decrease in the rate of acid production should be predictable from the reduction of oxygen concentration in the mine as determined from analyses of the mine atmosphere. To determine this, copper tubing was inserted through the stoppings in several sealed mines, so that atmosphere samples could be withdrawn from a point well back of the seal. Four of these mines were comparatively shallow-cover drift mines delivering acid water. The fifth was a shaft mine with a minimum of approximately 160 ft of cover. Samples were taken and analyzed by the customary gasometric method. The results are shown in Figs. 2 to 6.

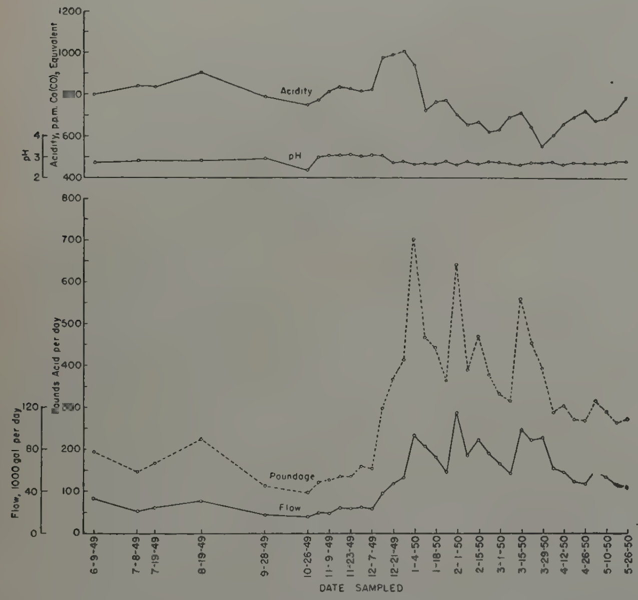

Fig. 10—Fluctuation in pH, acidity, flow and pounds of acid per day of Mine D.

It will be noted that there are fluctuations in oxygen content in most of the mines, but the maximum oxygen reduction in any of the drift mines is to about 11 pct, with frequent returns to substantially pure air, whereas in the shaft mine the oxygen decreased to 1.5 pct with a maximum of return to 8 pct. The observed oxygen content in the shaft mine is comparable to the oxygen percentages observed when deep mines are sealed to extinguish mine fires. It is not known whether such data have ever been obtained before on shallow-cover drift mines.

Mines A, B, C, and D were also intensively studied by weekly sampling of their discharges. These mines were comparatively small workings isolated from other mines, which eliminated possibilities of interconnection. They were mines that were long abandoned, and three of them had been sealed previously but no maintenance had been given the seals.

The drainage of these mines was sampled weekly regardless of weather conditions. Samples were taken in completely filled bottles that were kept below mine temperature, approximately 56°F, until subjected to analysis. Acidities were determined by titrating in hot solution to the phenolphthalein end point. Nearly neutral or alkaline waters were first acidified with standard sulphuric acid heated to boiling and back-titrated with standard alkali. The rate of flow from the mine was measured by means of a V-notch weir as the samples were taken for the total acidity and flow. From these two factors could be calculated the actual poundage of acid delivered by the mine to the receiving stream.

The mines on which no work had been done since abandonment were first dewatered by removing the entry stoppages caused by falls and cave-ins at the outcrop. The drainage then was sampled long enough to determine the effect of free flow. In the case of mines that had been sealed previously, the seals were removed and as much impounded water as possible drained out. These mines were then resealed, as was one of those that had never been sealed before. Figs. 7 to 10 illustrate the general activity pattern of these mines. They show the pH, acidity, flow, and pounds of acid delivered per day based upon regular weekly sampling, the samples being taken not only on the same day in each week but at substantially the same hour. During the summer or low-flow period the acidity is slightly higher than during the later high-flow periods. The most striking factor is the relationship between flow and acid poundage. These curves parallel each other closely, indicating that acid delivery is almost directly proportional to flow, whereas the acid concentration or acidity, while showing considerable variation, is in a sense relatively constant.

The curve for mine A, Fig. 7, shows the beginning of increased flow on November 23. At this time the poundage and acidity also increased. On December 14 the flow reached a crest with a slight drop in acidity, but a tremendous increase in pounds of acid delivered. The amount actually delivered on November 23 was 30 lb, on December 14, 735 lb, an increase of 2350 pct, while the flow increased from 8000 gal to 175,000 gal, or 2087.5 pct. Furthermore, instead of the acid becoming diluted, its concentration increased from 425 to 525 or 23.5 pct. The next crest, on January 4, shows a decrease in acid poundage from the previous crest of 6 pct, with an increase in flow of 70 pct and a decrease in acidity of 33.3 pct. The third crest on February 1 shows a further increase in flow over January 4 of 19.2 pct, a decrease in acid poundage of 8 pct and a concentration decrease of 33.3 pct. This may be attributed to the flushing action of the high water, since it reached areas from which no water had been flowing over a period, and hence carried out acid produced but not delivered during the low flow periods. Intermittent flushing in this manner has the apparent effect of decreasing acid production. It should be noted that during all those fluctuations the pH remained almost constant, the variation being from 2.58 to 3.04, again demonstrating that pH measurement does not indicate many of the important properties of acid mine drainage.

The other mines follow the same pattern, the discrepancies being a difference in degree and not of kind. Since the flow is the most important effective variable, it is plotted on a single graph to illustrate the general pattern. The curve for mine D, Fig. 10, is erratic prior to November 4, as sealing operations were in progress and natural flow was impossible. Subsequent to this period, the four mines follow the same pattern of flow with the exception of mine C, which does not show a crest on December 14, but the increase in flow is gradual to the crest on January 4.

During the opening of mine B, Fig. 8, it was evident that large amounts of acid were being released, and since the receiving stream was stocked with trout and was at low flow, samples were taken to determine the effect on the stream and to stop the addition of the excess acid if a danger point was reached. This condition actually arose, and the flow of acid had to be halted to permit recovery of the stream, after which drainage was allowed to flow only at such a rate as to maintain the proper alkalinity in the stream.

The study of the effect of acid entering the stream was continued by means of weekly samplings to the completion of an annual cycle. The curves in Fig. 11 illustrate the results of this study on a pure mountain stream receiving the acid from two mines, A and B, described above. The curves represent the alkalinity of the stream at a point 6/10 of a mile above entry of the acid and 6/10 of a mile below the entry of the acid, and designated as before and after acid entry. The third curve represents the total pounds of acid per day entering the stream from mines A and B.

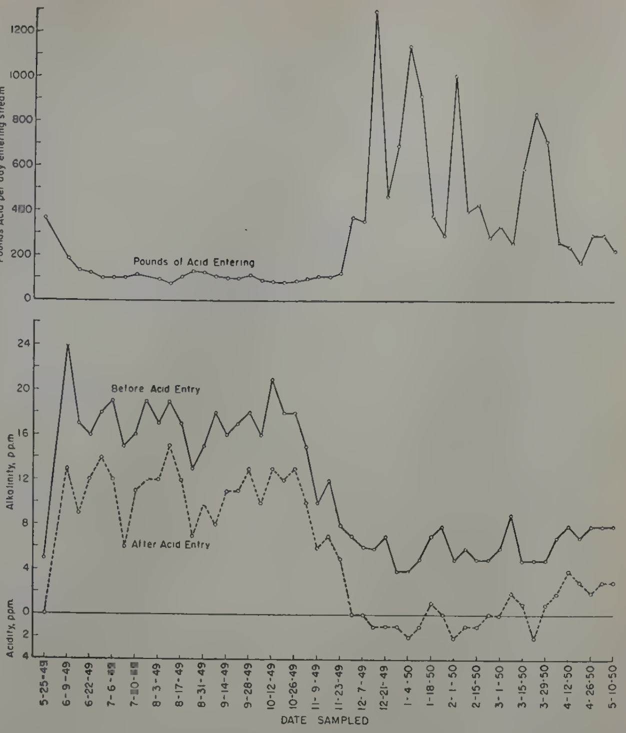

Fig. 11—Effect of acid delivered from Mines A and B on the receiving stream.

These data indicate a constant reduction of alkalinity of the stream irrespective of the amount of acid it receives. This may be explained by concurrent fluctuation of stream flow and mine discharge.

Also of note is the high alkalinity of the stream during the summer and fall months, the low-flow period, as compared with its alkalinity during the high-flow period, winter and spring.

Conclusions

1—Acid is produced by the oxidation of sulphuritic iron compounds in and associated with the coal measures. This reaction probably is a normal chemical reaction for which an equation can be written.

2—The amount of acid delivered by a mine is not the total amount produced in the mine but rather that accessible to the transporting agent, the volume of water flowing through the mine.

3—pH measurements in no way represent either the concentration or poundage of acid delivered or produced by a mine.

4—The delivery of acid from mines follows regular patterns, varying in degree and not in kind, when sampling intervals are short and are sufficiently continuous to establish the pattern. Since they deal with water flow that is dependent upon precipitation and ground water levels, they fluctuate with seasonal and daily precipitations. Annual, semiannual or quarterly sampling lead only to highly erroneous conclusions.

5—The production of mine acid is a process of nature, and while it operates on all coal measures and their associated strata, it is increased materially by any disturbance of geological formations that results in exposure of the reactive constituents to air and water.

The History and Development Of Phosphate Rock Mining

by R. B. Fuller

DURING the summer of 1949, the United Nations Scientific Conference on the Conservation and Utilization of Resources met at Lake Success. As summed up by one writer, the purpose was: "That everyone will try to learn how to feed more people from an acre of land." That objective is a stimulus to the phosphate rock industry as well as other industries.

Phosphate rock was mined first in the Canadian province of Quebec where it was found as an apatite in pockets. In 1863, mining began near the Rideau Canal, first in open trenches and quarries until shafts were sunk. These operations were difficult since hard rock had to be drilled and blasted. Steam was the source of power. The phosphate rock was hand-selected and recovery ranged from 6 to 10 pct, averaging 70 pct to 85 pct bpl (bone phosphate of lime). Costs, f.o.b. Montreal, were $14 a ton, and selling prices averaged $17 a ton. In 1885, about 29,000 tons were sold on this basis, but by 1892 the output fell to 8000 tons at an average price of $15 a ton at Montreal.

South Carolina Deposits

Phosphate rock was first discovered in South Carolina in 1837, but a mining company was not formed until 1867. It was immediately successful, and the industry continued to develop and ship an increased tonnage of land and river rock, shipments increasing from approximately 20,000 tons in 1868 to 537,000 tons in 1890, when the shipments began to decrease. In 1892 only 350,000 tons were shipped. In this field there were two types of deposits: land rock and river rock. More land rock tonnage was mined until about 1887 when river rock production began to equal it. The land rock was mined adjacent to and in the river marshes by pick, shovel, and wheelbarrow. Each worker dug daily a space 6 ft wide, 15 ft long and 6 ft deep, which was overburden, and then loaded about 3 cu yd of phosphate rock into wheelbarrows. His helper would wheel the barrows to small steam railroads up to 250 ft distant. Water was kept from the pits by steam pumps connected to railway cars.

The phosphate rock was loaded into railroad cars, hauled and dumped into a log washer to which water was added, and run over trommel screens to eliminate the clay and loose sand. Steam shovels for loading were introduced about 1891.

River rock was mined by floating dipper dredges and at low tide was loaded into barges by laborers using oyster tongs, the large lumps being hand-loaded. Barges and bateaus were unloaded by wheelbarrows which were wheeled to the same type of land rock washers. The washer product was piled with stacked pine cordwood in alternate layers and fired for drying. Usually this was done under sheds along the river bank. The dried rock then was transferred to ships, usually three or four-masted schooners.

Cost of production of both land and river rock, including $1 a ton royalty paid to the state for state-owned property, was approximately $4.25 a ton averaging 60 pct bpl. The selling price was approximately $7 a ton f.o.b. In 1886 the courts handed down a decision permitting anyone to mine, and many additional companies were organized immediately. As a result, the selling price was lowered to a profit margin of 50¢ or less per ton. The South Carolina mines were eventually shut down when a more profitable source of phosphate rock was discovered in Florida.

Florida Deposits

Phosphate rock was discovered in Florida in 1882, and in 1883 a small quarry was opened near Hawthorne. In 1886 phosphate rock of good grade was discovered on Peace River, and the Arcadia Phosphate Co. was formed. In 1888 the company began shipments amounting to 3000 tons that year. During the latter part of 1888, hard-rock phosphate was discovered near Dunnellon. Exploration extended rapidly, and deposits of hard-rock phosphate were mined over a wide area so that shortly after 1900

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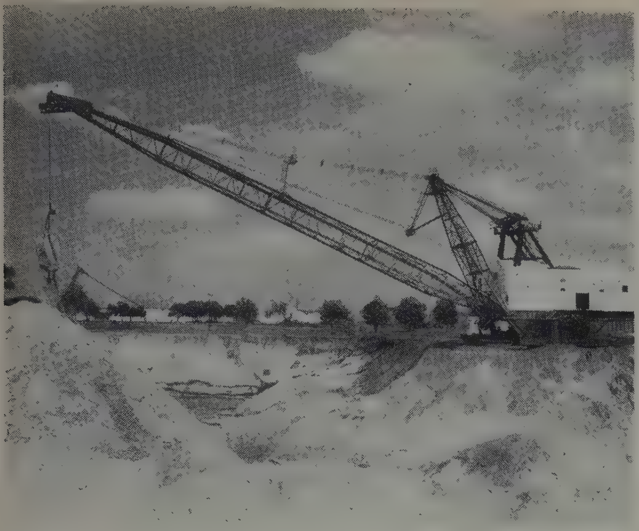

Fig. 1—The Bigger Digger, Bucyrus-Erie 1150-B dragline, 215-ft boom, 21.4-cu yd bucket.

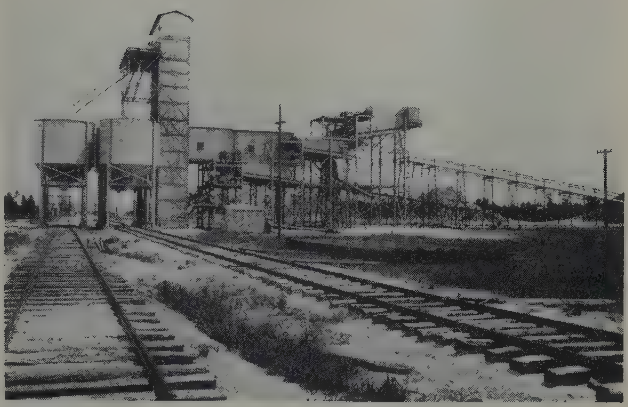

Fig. 2—One of the first all-steel washers, 1928.

the census bureau reported 81 companies mining phosphate rock in Florida. Centrifugal pumps were used in the mining of Florida river pebble rock. They were 8 in. or 10 in. in size and were installed on a dredge with steam engines and boilers, which used wood as fuel. Alongside the dredge was a washer barge which had the usual trommel screen for discarding any tree roots or other foreign materials more than 1½ in. in size. Following the trommel were log washers and a rotary screen for rinsing. The washer rock was discharged by a chain drag into small barges, which were towed to the drying plant on shore and unloaded by shovels into an elevator or wheelbarrow. The rock was dried in 48-in. diam by 30 ft long rotary kilns using wood as fuel. The cost of dry river pebble loaded in railroad cars or lighters was reported to be $1.75 a ton, with a selling price equivalent to $5.50 a ton for 60 pct bpl.

About 1890, however, there was further activity at the headwaters of both the Peace and Alafia Rivers, and many new companies began mining principally in the creeks and shallow ponds adjacent to railroads. Thousands of acres originally sold as timber lands to Hamilton Disston by the state for 25¢ per acre were afterwards found to contain high-grade phosphate deposits.

Lower cost water transportation probably allowed the river pebble miners to compete so successfully with the South Carolina miners. The river pebble when dried on the lower extremities of Peace River was loaded into large lighters, which were towed to the entrance of Charlotte Harbor. The schooners had sheltered anchorage inside of the pass and would take on large portions of the cargo from the lighters with ship's crane. However, to get a full load the lighters and the ship were towed to deep water beyond the harbor where the cargo was transferred from lighters to ships, which delivered the cargoes both coastwise and to foreign ports. Eventually, a railroad was constructed from Arcadia to Punta Gorda and afterwards from Arcadia to Boca Grande.

Wood for fuel was becoming scarce and more expensive, and eventually some of the miners and the railroads brought in coal, and a coal storage dock was constructed at Boca Grande. About 1892, production in the United States was about evenly divided, approximately 350,000 tons each from South Carolina and Florida. Hard rock mining had been further developed in the northern and north central part of Florida. At first the rock was strip-mined, sorted by hand, and the selected lump phosphate dried in a pile on wood. The deposits were of a pocket type and did not occur in a blanket form over extensive areas as in the pebble field to the south. This was a lucrative business requiring little capital. The grade of rock was 80 pct bpl plus and was shipped mainly through Jacksonville and later through Fernandina.

Most of the capital for earlier pebble companies came from foreign sources. It was not until World War I that the French interests sold the last of their holdings because of the discovery of high-grade phosphate in Morocco.

The improvements in the hard-rock field followed the usual pattern. First, dredges were floated in pits and the excavated phosphate treated in log and screen washers. More fines were saved, but there was still a substantial loss. The washers were improved and the operations thoroughly mechanized, the fines being saved by tabling.

A central loading, storage, and drying plant was located at Fernandina, an export point until recently when a modern drying and storage unit was installed at Dunellon. Walking type draglines now handle the overburden efficiently.

With the exhaustion of river pebble deposits, which were placer deposits to a great extent, and the discovery in the Bartow-Mulberry area of more concentrated and higher grade deposits, many new mining companies were formed, which later were combined into a few. About this time electricity came into more general use.

Meanwhile a great many of the hard-rock pocket mines were mined out. Their costs were increasing so that by 1905 most Florida phosphate was mined in Polk County, or South Florida. Most creeks and shallow rivers had been dredged out, and the land pebble miners were forced to remove deeper overburden. This was started with mules and scrapes, and later when overburden up to 20 ft could be profitably removed, by steam shovels, dump cars, and steam locomotives. A great deal of overburden removal was contracted for at rates ranging from 12¢ to 25¢ per cu yd.

After the overburden of 15 to 20 ft had been re-

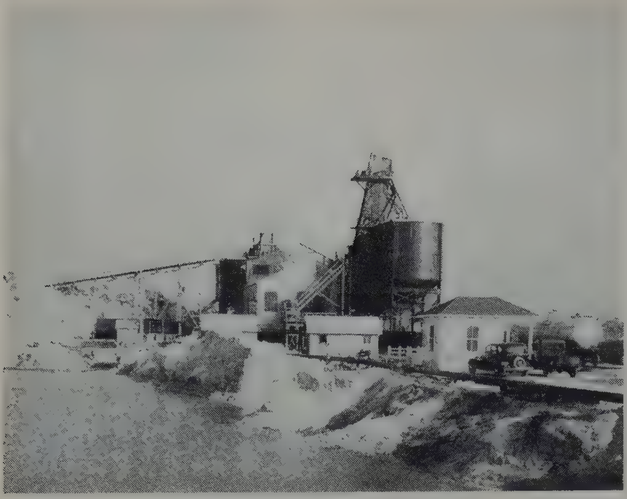

Fig. 3—Phosphate flotation plant constructed for recovering phosphate from formerly discarded washer debris.

moved, the matrix was sluiced with clear water to the suction of a 10-in. centrifugal pump by hydraulic giants operating at the rate of 2500 to 3000 gpm at a pressure of 125 lb. These centrifugal pumps were belt-driven by steam engines located 30 to 40 ft from the pumps; the boilers for the engines customarily were placed on top of the bank. The water for the hydraulic giants as well as for the washer was furnished by artesian wells 10 in. in diam sunk to depths varying from 300 to 600 ft. An air lift was used to get the water to the surface. The wells were located at central pumping stations and the triple-expansion, steam-driven plunger pumps or crank and flywheel pumps with fairly high vacuum had an initial pressure at the pump of 180 to 225 lb. However, in piping this water to the hydraulic giants, there was tremendous friction loss and a leak at practically every joint of pipe. Further developments in the centrifugal pump and the broadening of the use of electricity eventually eliminated some of these inefficiencies.

Several companies operated gas-producing engines using wood and coal as fuel, while others put in diesel engines. One installation had eight very large Busch-Sulzer diesel engine generators and Worthington-Snow diesel engine generators. At that time most of the phosphate companies were using wood as fuel for drying. One company used 600 cords of wood a day.

To meet the competitive prices of phosphate rock then prevailing, larger capital investments were required to lower the cost and make for a more continuous and uniform operation.

Where steam engines had formerly been used for operating the washers, the advent of electricity soon made a change, and motors were substituted from 1907-10. In practically all cases the motive power was transmitted to a central shaft and thence by means of counter shafts, gears, and bevel gears to the various moving parts of the washers. The general flowsheet of a phosphate rock washer was as follows:

The 10-in. centrifugal pump discharged the phosphate rock slurry through similar sized pipes to an entry box about 45 ft above the railroad track. The box had gates arranged to allow the mass of matrix containing 15 to 20 pct solids to pass onto sloping flat screens, which had staggered slotted openings of 3/64 x ½ to ¾ in. long, the surface area being governed by the percentage of solids in the matrix.

Large quantities of sand and disintegrated clay and fine phosphate rock particles escaped through the opening in the screens and went to waste with the water. The remainder of the material was swept into a horizontal trommel screen in which were inserted spiral flights for moving the +1½-in. material to a launder that carried these larger particles to waste. Under these trommel screens there was a similar screen which conveyed the material with additional clear water to customary log-type washers. From the washers the material went either to a shaking flat screen actuated by an eccentric or to rotary screens with the same size openings. It was customary to put this product through another set of log washers and then onto a dewatering screen before elevation into the bins under which cars were spotted for loading and shipping to the drying plants.

All the mining structures at that time were built of first-class pine and cypress timber, cut and manufactured from the area to be mined. Limited electrical power and the necessity for short pumping distances usually caused the mining plants to be located in the center of 200 to 240-acre deposits, which ordinarily were mined out in about 8 to 10 years. It was customary at the time for contracts to be written on the monthly average grade of the average analysis of the daily shipments. If waterborne, there were no particular stipulations as to the grade except a standard minimum, customarily 66 pct bpl, with premiums paid for excess grade.

Shortly after 1907, however, prospecting revealed that certain stratas in certain areas were consistent as to quality, and competition being more keen, contracts were made which specified minimum grades of 66, 70, 72, 75 and 77 pct bpl. About that time a more diligent search for high-grade possibilities was made by the existing companies and new companies then being formed. The steam turbine was introduced and proved a more economical and better source of electrical energy. New mines started up, which were entirely electrified except for the pressure pumps furnishing water to the mine.

Labor costs were $1.50 per 12-hr day. Many of the mines operated 24 hr a day, 6 days a week.

Another change in mining practices put into effect at this time was the hydraulic removal of overburden. This method required large vacant areas in which to impound the overburden and save the water for further use after clarifying. As later developed, it was unfortunate that a great deal of this overburden was dumped into the same areas that were also receiving the washer rejects. From then until World War I, both the Florida hard-rock and land pebble rock miners, in spite of keen competition, managed to survive and had practically put out of competition the South Carolina operators. Most of the high-grade Florida phosphate rock was exported and brought premium prices in Europe, South Africa, Australia, and New Zealand.

Improvements followed rapidly in Florida. Steam draglines replaced the hydraulic method in hard pan areas. Larger and more efficient draglines were developed, and finally the modern 21-cu yd machine, which greatly reduced stripping costs, was installed.

Mining methods were improved by moving the mine pump pit car out of mud and floods of pit bottom. The big dragline both stripped and mined the matrix and piled it in front of the pit car where it was washed into the suction sump with low pressure water. Before, high pressures of 200 psi or more mined the matrix from the face.

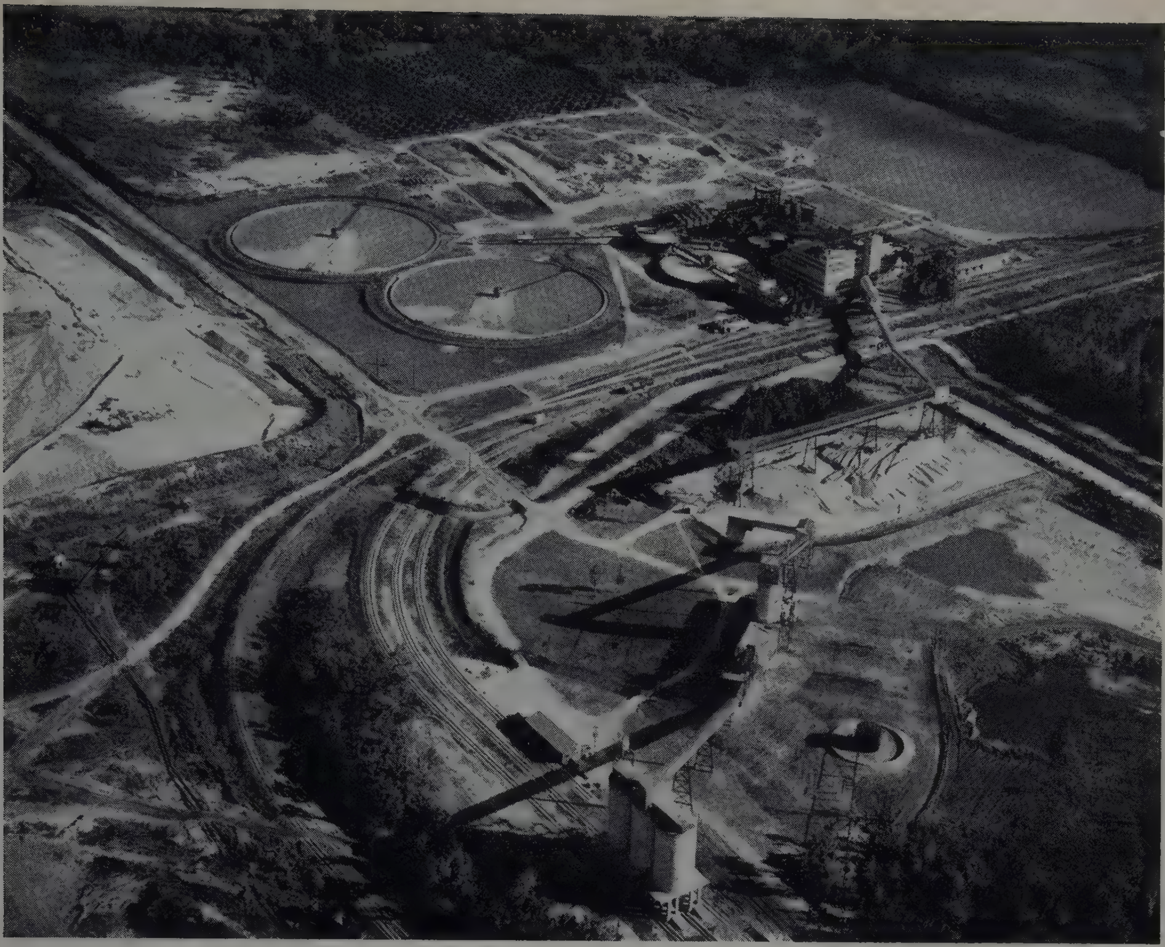

Fig. 4—1949 thickeners, flotation plant, conveyor system, and wet storage and drying facilities, Noralyn, Fla.

Immediately after World War I began, practically all of the operators in Florida shut down for several months, but when it became evident that the United States would be involved, the Department of Agriculture and those affiliated conducted an extensive educational program for greater crops. After World War I the producers of phosphate rock went into a depression because it was several years before definite plans were worked out for the rehabilitation of the soil of the devastated European countries. It then was realized that the Florida phosphate fields had lost a considerable portion of its foreign markets to producers in the Pacific and Indian Ocean Islands and that the new African deposits were becoming more extensively competitive in the European field.

During this time of curtailed activities, the companies that remained, about three hard rock and ten land pebble phosphate rock operations, did extensive research work with a view to making greater economies by increased use of electricity and by mechanical improvements. Two of these, the Bigger Digger, and one of the first all-steel washers, are shown in Figs. 2 and 3.

It was discovered that beneficiation by the flotation process could be adapted to waste products that consisted of large quantities of phosphate particles and that these fine sizes could be recovered economically. The first pilot flotation plant was constructed in 1927 by the predecessor of International Minerals and Chemical Corp. The application of the flotation process for recovery of phosphate is the most noteworthy achievement in the industry since phosphate rock was first mined on this continent.

At one mine the overburden had been stripped and hauled away in tram cars for a period of years, leaving a large mined-out area from which there had been recovered slightly less than 1 million tons of all pebble rock with an average grade of 75 pct bpl. The washer reject, or debris, was dumped into the mined-out pits adjacent to the washer without contamination with overburden. When the original washer had been abandoned, a flotation plant was constructed on the old washer site, and from this debris, which was pumped to the flotation plant with a floating dredge and centrifugal pumps, there were recovered over 1¼ million tons of 77 pct minimum bpl. A phosphate flotation plant constructed for recovering phosphate from formerly discarded washer debris is shown in Fig. 3.

However, flotation has made necessary much larger capital investments for the establishment of a mine and plant. It now has become necessary to make investments in the most up-to-date machinery and equipment and to increase in size and area any producing unit in order that profits may be realized and a higher grade produced and sold at prices comparable with the selling price 50 years ago. This has reduced the number of operations in Florida to one hard rock producer and eight land pebble rock companies. Today labor costs are more than four times what they were 20 years ago. Other items such as fuel, supplies, etc. have increased in price, yet even with technical developments, more labor has been employed. There is three times as much output per man hour as 20 years ago.

The greatest effect that the flotation process has had in the Florida field is to prolong the life expectancy of the industry and to increase the known possible and probable reserves, thus warranting the increased capital investments. A modern flotation plant and conveyor system are shown in Fig. 3.

Through the next several years there no doubt will be scientific developments that will lead to further recoveries, extending the life of these deposits and causing an ever-increasing tonnage of phosphate rock to be shipped from Florida.

It is interesting that during World War II extensive shipments of phosphate rock were made from Florida to Buckingham, Quebec, which is within 30 miles of where phosphate rock was first mined on this continent. It seems significant that when the United States entered World War II, the initial attacks were made at points which quickly gave control of the phosphate deposits in North Africa and the Pacific Islands.

Tennessee Deposits

The Florida producers were confronted with competition in the domestic markets when in 1893 "blue" rock was discovered in Hickman County, Tenn.

Many blue rock mines were started, first to mine the accessible outcrops and then to follow the rather thin seams of 30 to 36 in. thickness underground, at increasing costs. One early mine at Toomey, well equipped with the then modern air drill, tram cars, crusher, and dryer survived a number of years but many other operations were unsuccessful. One operation at Gordonsburg operated very successfully until well into World War I. It was well equipped, and costs at first were quite reasonable, but thinning deposits that dropped lower in grade finally resulted in abandonment of the whole project.

White rock phosphate in Perry County had a similar history. The deposits were extremely spotty and erratic, but some high-grade phosphate was mined. During World War I, there was a brief revival of mining, but it could not survive competition.

Brown rock phosphate operations were started about 1896 in Maury and Giles County, Tennessee. At first, only select high-grade lump was stripped and mined by hand. The clay and fines were removed by hand screening or shaking out on a ten-tined potato fork. The lump was sometimes sun-dried on a "yard" or in bad weather stacked on layers of cord wood, which were burned to dry the phosphate.

About 1908–09 simple type washers followed by screens came into use to work the muck or underlying finer phosphate after the lump had been mined. Wood and coal-fired rotary driers replaced drying on wood for the fines.

About 1910 to 1911 progress started with the introduction of Dorr rake classifiers and thickeners for better washing and to save more of the fines. Hydroseparators followed Allen cones.

Among the Tennessee firsts was the use of a wet cyclone in 1923 and the first research on Tennessee phosphate flotation in 1927. Steam shovels were tried early in Tennessee, but the dragline introduced in 1914 gradually replaced mule scrapers and the like. Steam tram line transportation along with mule team haulage survived until about 1941, when the mule team faded from the scene, and as the larger blanket deposits were mined out, the truck took over haulage from the mines. The diesel dragline replaced the slower steam machine.

About 1936, the Tennessee high-grade deposits were being rapidly exhausted. At the present time little so-called acidulating or high grade, now 70 pct bpl, is being produced, but two factors came in to maintain Tennessee's second place in domestic production.

The consumption of 65 pct bpl phosphate for ground rock for direct application increased, and demand for electric furnace feed of still lower grade grew rapidly.

Now phosphate as low as 55 pct bpl is electric furnace feed; this permits the use of almost all the remaining reserves with blending and beneficiation by washing or similar treatment. There are many years ahead for the Tennessee phosphate industry.

Western Phosphate Reserves

Since recognition of phosphate in the West near LaPlata, Utah, in 1889, many enterprises have been launched, especially during World War I. Several underground mines were opened along with plants for crushing, drying, and grinding. After the war cheap water rates from Florida to the West Coast were restored. Florida captured much of the Western phosphate market and caused some of these early operations to shut down. Western phosphate reserves are largely in steeply dipping underground deposits of lower grade than Florida high-grade pebble. When conditions are favorable, it is possible to obtain a mining cost of $2.50 to $3 a ton, but more often the costs will be much higher.

The Anaconda Copper Co. has a successful mine in the West producing 68 bpl rock by top slicing in a fairly thick vein. The rock is crushed, dried, and shipped to Anaconda for producing triple superphosphate with the use of byproduct sulphuric acid.

There is another active underground operation near Garrison, Mont., where the output goes to Trial, B. C.

At Sage, Wyo., and Hall, Idaho, there are two strip mines where phosphate rock is being produced in considerable tonnage at very reasonable cost to supply the West Coast demand for fertilizer and for electric furnace operations at Pocatella, Idaho.

Some of the largest known reserves of phosphate rock are in Montana, Idaho, Wyoming, and Utah, and these deposits will be increasingly exploited with the expanding demand in the West for plant food.

From the Bureau of Mines, United States Department of the Interior, Mineral Market Report No. 1761:

"A new high record was made in the mine production of phosphate rock in the United States in 1948, according to reports submitted by the producers to the Bureau of Mines, United States Department of the Interior, the total reaching 9,388,160 long tons, more than quarter of a million above the 1947 record high. Of this, 7,184,297 tons were mined in Florida, over 800,000 tons greater than in 1947; 1,499,547 tons in Tennessee, only slightly more than in 1947; and only 704,316 tons in the Western States, about half a million tons less than in the previous year."

The average value of the phosphate rock sold during the first half of 1948 was $5.70 per long ton.

Scientists are conducting experiments which no doubt will lead to more efficient and more widespread use of phosphate compounds.

For the present there are two principal problems that warrant careful and immediate study and correction: 1—Cheaper power and 2—a lowering of the cost of distribution of the Florida phosphate rock. Florida has a bountiful supply and the world needs ever-increasing quantities.

Developing Mesabi Orebodies Under Lake Beds

by James R. Stuart

As the available remaining properties of iron ore reserves on the Mesabi Range are opened up for mining, the various properties located under lake beds are brought nearer an active status. The actual physical problems involved in stripping these properties do not act as a deterrent so much as the legal and political problems that are encountered.

When it is proposed to destroy a natural lake that has been used by the public for many years, much local as well as state opposition may be encountered regarding its destruction. Public hearings must be held and some adverse publicity is likely to result. The ownership of the ore under the lake and the rights of the abutting property owners must be settled, and protection from damage caused by a disturbance in surface and subsurface drainage is likely to be demanded by property owners some distance from the proposed mine area.

The Embarrass Mine, located near Biwabik, Minn., falls into this classification. A portion of the orebody lies under what was formerly Syracuse Lake, this body of water having been removed in the process of stripping the mine.

An additional problem in the case of a meandered body of water is the establishment of a meander line that can be projected downward as mining progresses to form the basis for a satisfactory division between lake bed and upland ore shipments for royalty purposes. Fig. 1 illustrates the complications encountered in maintaining these divisions. A balance point was agreed upon in the center of the lake to make an equable division of lake bed ore to the abutting properties. The entire lake bed has since been adjudged the property of Minnesota.

Lake Characteristics

Lake bed stripping problems with which this paper is concerned necessarily are limited to a specific type of lake, namely the glacial lakes of the Lake Superior region. One characteristic common to these bodies of water is a deposit of fine black mud or silt on the bottom, frequently underlain by a layer of impervious blue clay. This is also true of the muskeg areas of the region, which present almost identical problems as lakes in stripping.

The actual removal of the water and the lake bed material is a routine matter more or less standardized as to equipment, and the period of time required can be estimated easily on the basis of volume and capacity. More important than the foregoing is the execution of preliminary work, and above all, the timing involved. An account could be prepared based entirely on statistical and cost data which would give a very fair picture of the time required and cash outlay needed to effect the removal of a body of water preliminary to stripping the orebody. However, the real interest from the standpoint of the operator and the engineer who carry responsibility for completion of the job lies in the unexpected emergencies and the action of various materials involved in the stripping when the balance has been upset through diversion of water courses and the reduction of the lake level.

Runoff and Drainage

Lakes are located in natural basins that catch all the rain water and runoff water for a considerable area. Where a lake is involved having an inlet and outlet or a sizeable water course running through it, the drainage area may include a watershed covering many square miles. All available data then must be collected to supply a history extending over as many years for which information can be gathered on the flow of streams, annual rain and snowfall, and most important, the peak flows to be expected. Where the diversion of a stream around the stripping area is a part of the problem, this last factor is of great importance since it controls the cross-section to be selected for the diversion channel and the volume to be removed in its excavation, as well as affecting the hydraulic considerations to be met in the design of the completed channel.

Characteristic material in the overburden found at the Embarrass Mine is illustrated in Fig. 2.

Well Pumping

Pumping from the well holes was started well in advance of the draining of the lake. Fig. 3 shows a gradual lowering of the water table with no noticeable fluctuations during the period in which the lake was being dewatered. Unfortunately, because of tight ground, a maximum flow to the wells was not maintained. This retarded the rate at which the water table was reduced so that in the course of stripping the excavation soon extended below the water table, and the great bulk of the pumping was handled from a system of sumps in the pit itself. Any dewatering program projected by prepumping from wells, a glorified well point system, would have to be started well in advance of the stripping to be of any great advantage.

Preliminary drainage of the surface over the mine area is entirely apart from the actual elimination of the lake bed itself. Since the lake is what is called a perched water table because of the impervious character of the lake bottom, the adjoining surface may be dewatered below the surface of the existing lake and the flow will not be affected by the proximity of that body of water. This condition actually has been demonstrated through the establishment of a number of observation holes where a small churn drill was used to put down the holes and a 3-in. pipe was installed for taking water level

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This page of Transactions is a continuation from p. 712. The missing

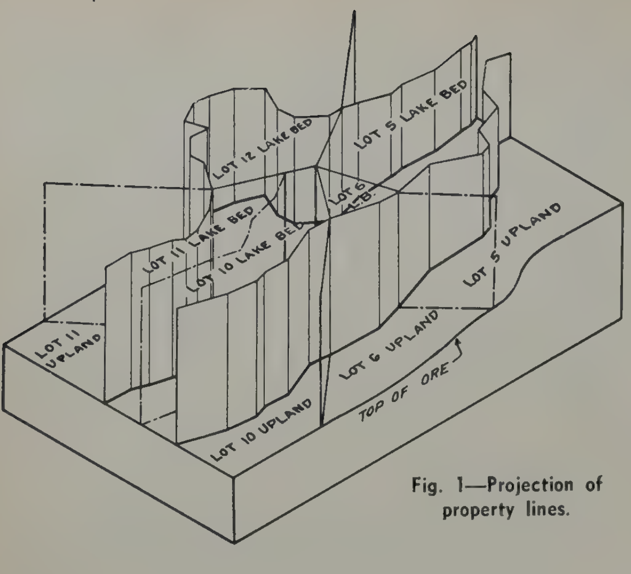

Fig. 1—Projection of property lines.

measurements. Water was pumped from a number of large diameter well holes put down with a rotary drill rig and using a bit diameter of 32-in. Deep-well pumps were used in dewatering from these holes.

Under natural conditions the surface of a lake is the water table. From the contact of the water with the shores, the water table in the ground slants upward roughly paralleling the surface. The first efforts to control drainage in the vicinity of the pit would be the establishment of ditches outside the perimeter of the proposed stripping area to catch surface runoff from rainfall and natural drainage and conduct it away from the mine.

A generalized cross-section showing effect of well and sump pumping on the water table is shown in Fig. 4.

Diversion Channels

Returning to the problem of stream diversion, construction of the diversion channel in itself tended to reduce the elevation of the ground water on that side of the pit area, since there was a flattening of the water table between the mine and the channel. A disturbing element here was the possibility that a pervious gravel layer might be cut by the channel, allowing the diverted water to enter the ground as a subsurface flow and find its way to the stripping area. However, no indication of this condition appeared in the pumping record after the stream was diverted.

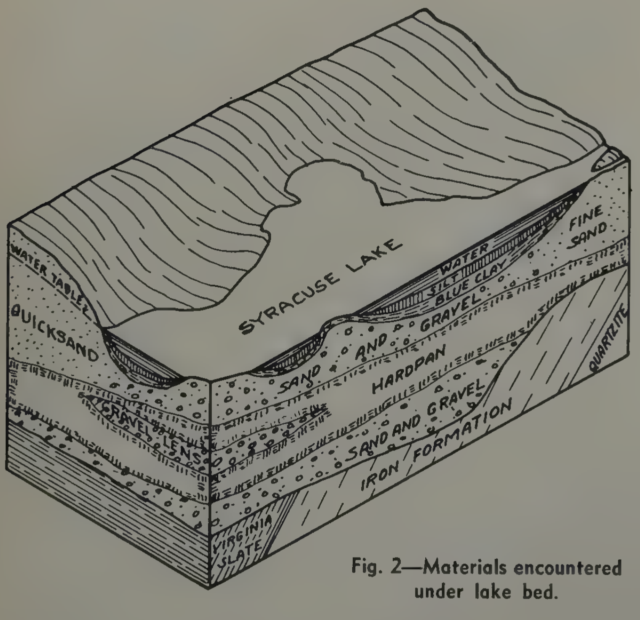

Fig. 2—Materials encountered under lake bed.

As is true of many projects where climatic conditions are changeable and winter operating conditions severe, the timing of various projects eventually got out of step. Dewatering of the pit area was accomplished through the use of sumps. It became necessary to enlarge the pit area by dewatering the lake, and this in turn required a temporary diversion of the stream prior to completion of the diversion channel. A temporary diversion channel therefore was constructed along the east shore of the lake to accommodate the flow and allow dewatering of the lake. No noticeable leakage developed from this temporary channel into the pit area while it was in use.

The steps in water diversion before completion of permanent diversion channel are shown in Fig. 5.

Dewatering Lake

Since the lake had a maximum depth of only 20 ft, the head on the dewatering pump was not a problem. A centrifugal pump with a capacity of 3500 gpm was installed on the dike between the lake and the temporary diversion ditch for this work. The same pump was shifted downward as the suction lift increased. The overall period required to pump out the lake was 19 days.

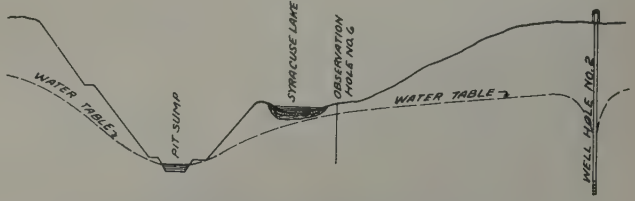

Fig. 3—Generalized cross-section showing effect of well and sump pumping on water table.

Ordinarily at this stage of the operation with the entire lake bed exposed, the supposition would be that the whole lake bottom, in this case approximately 30 acres, would present a surface of semifluid black mud. As it developed this condition was modified by the fact that the lake water, as it was pumped out, relieved the pressure against the bottom mud. When this balance of pressure was disturbed, the lake bottom mud started to creep and flow and followed the water to the low point in the basin. This fortunate tendency resulted in providing a stripping area of the original lake bed that contained a shallow layer of lake bottom mud that could be mixed with the glacial material below and was loaded out as normal stripping. A part of the deeper mud was bailed out by dragline and removed with bottom dump trucks. It was found that as the winter weather became colder, and subzero temperatures were the rule, the lake bottom material became very much less fluid, thus expediting its removal.

Subsurface Conditions

The condition of the glacial materials, both sand and gravel, directly under the lake bottom mud was very dry—definite proof that the lake water had not fed into the subsurface drainage of the area.

This again was a reassuring condition since deeper lakes were located both upstream and downstream from the mine. Leakage from either one of these bodies of water would have intensified the stripping problems.

A point to be noted was that in spite of the fact that the entire lake bed was stripped out and the elevation of the bottom of stripping was carried well below original lake level, the sump being 100 ft below this elevation at this time, the entire flow of the river was carried in a temporary diversion channel through the winter at the toe of stripping until it was turned into the completed permanent diversion in the spring of the year.

Quicksand Problems

A major feature in the early stages of stripping at the Embarrass Mine was the large area of quicksand that extended to a considerable depth before hardpan stripping was encountered. Lowering of the active stripping area required special treatment here since no means could be found to dewater the fine sand in advance of operations because of the steep gradient held by the water in the sand where a bank was opened up.

Power shovels were out of the question since the quaking sand offered no support for excavator or haulage trucks. Dragline equipment was substituted, and a main sump was established with a system of ditches leading into it. The initial lift was taken from the sump with the excavator standing on floats, and the material removed was cast back from the crest. This combination of sump and ditches dewatered a thickness of 5 to 8 ft of sand which was then loaded out into trucks by dragline, after which the process was repeated.

An operating problem of interest was the stubbornness shown by the quicksand when attempts were made to dig it with the dragline bucket. The saturated fine material was of such a livery consistency that when the bucket was pulled through it, water and very little sand was recovered. Bringing the bucket in on the ends of the teeth showed the same results. The technique finally developed was one of patience. The operator would let the bucket out to the limit of his reach, then allow it to sink slowly into the sand of its own weight before pulling it in filled with sand.

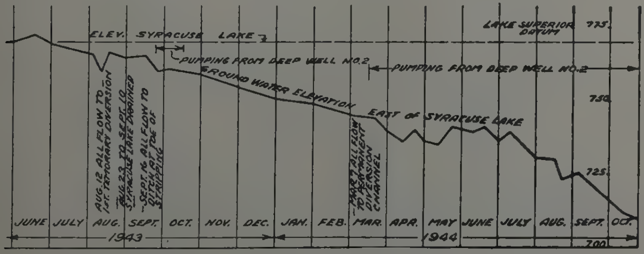

Fig. 4—Ground water elevations during diversion and pumping operations.

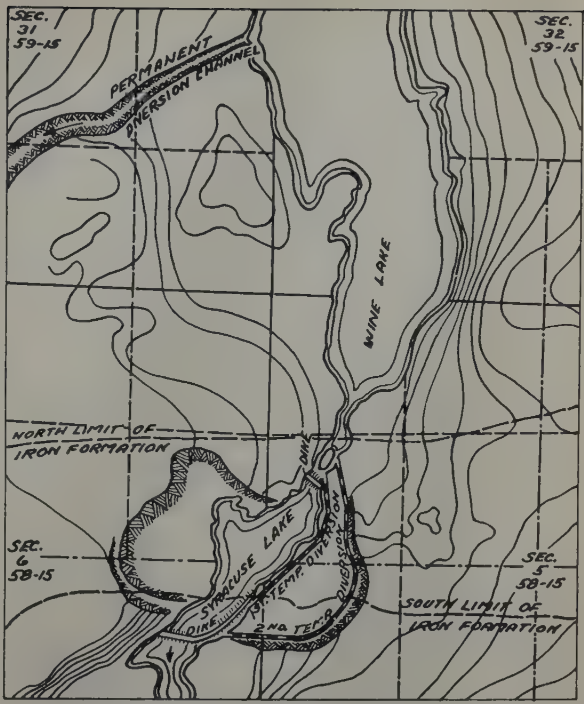

Fig. 5—Steps in water diversion before completion of permanent diversion channel.

The most satisfactory arrangement found for pumping from the sump was to mount the pump units on floats with flexible connections to the pipelines brought down the bank. In this way variations in water level did not endanger the pumping units, and in the event of power failure dewatering could be resumed quickly when power was restored. Since advance of the project was entirely dependent upon keeping the water down, pumping units were kept available at all times with a capacity of twice the rate of pumping required to hold the water elevation static.

Summary

The major considerations in developing orebodies that occur under lake beds would be:

1—Set up a schedule far enough in advance so that an orderly system of surface drainage controls would be operating prior to the start of stripping operations.

2—Where stream flow is to be controlled, removal of the channel to a permanent diversion location before operations were started would safeguard the work and would not tend to cramp the operation by forcing the use of a stripping plan that would make security of a temporary channel location a major premise.

3—The drilling of test holes to determine the type of material to be expected adjacent to the lake that is to be drained. This would serve two useful purposes. The holes would give a clue as to whether the ground could be predrained through the use of deep wells, and with a pipe casing installed, they would serve as observation points to measure the elevation of ground water during stripping and drainage operations.

The notes covered in the foregoing paper resulted from observations and experiences afforded by one particular mine, the Embarrass Mine on the eastern part of the Mesabi Range. They have of necessity been reduced to generalities that would apply to any similar project.

It is interesting to note that a pump operating at an average discharge of 3000 gpm would remove a volume of water equivalent to 7000 cu yd in an 8-hr shift. When the volume of a lake is regarded as displacing an equal volume of surface stripping, the simplicity of pumping, with no dump area necessary and a minimum amount of equipment and labor required, presents intriguing possibilities for economies.

• • • Adsorption of Silver Ion by Sphalerite • • •

by A. M. Gaudin, H. R. Spedden, and M. P. Corriveau

A preliminary study of silver ion adsorption by sphalerite in ion exchange column with influent electrolyte marked with radiosilver is described. Rapid ion exchange occurs at first followed by slow reaction controlled perhaps by solid state diffusion. One experiment with radiocopper instead of radiosilver is reported briefly.

IT has been known for many years that sphalerite is floated after it has been activated by a metallic cation, particularly copper.[1] The activation of sphalerite by copper is known to be an exchange of an atom of copper for an atom of zinc at the surface of the sphalerite.[2, 3] It has been estimated from various guesses of the available mineral surface that the depth of the copper-bearing coating is of the order of thickness of one atom. However, the exact mechanism of the activation process is unknown.

A number of metals other than copper act in the same general manner.[4, 5] In particular, it has been known that silver replaces zinc at the surface of sphalerite. As copper is usually divalent, whereas silver is monovalent, it may well be assumed that there are differences, perhaps major differences, in the detailed scheme of activation by these two cations. Furthermore, since silver has unusual photoelectric properties and may be a contributor to the fluorescence of sphalerite, a study of its activating mechanism might open up some interesting avenues for exploring these other phenomena.

The choice of silver as an activator of sphalerite was suggested also by the fact that its radioactive tracer, silver 110, is particularly convenient in that it has a long half-life and is energetic enough to permit counting through a counter window. The experimental technique selected, therefore, included the preparation of solutions of silver nitrate containing a proportion of active silver nitrate sufficient to give convenient counting data.

In further considering the method to be used, a desirable choice seemed the use of batch experimentation in which a given solution was allowed to act on a certain lot of mineral for a given length of time at a given temperature, this period of mutual reaction being followed by analyses of the separated solid and solution. But an alternative method seemed more attractive. This is the method now so familiar to the users of ion-exchange resins, which was first made famous by the Russian chromatographer, Tswett.[6-8] It consists in forming the mineral into a porous column through which the solution is allowed to flow at a constant rate. The usual method of allowing a batch of solid to react with solution is the equivalent of a single plate distillation operation while the chromatographic or ion-exchange column is a multiple-plate method—the multiplicity of the plates being represented by the reciprocal of the rate of flow of solution through the column. It seemed to us that the chromatographic method had such

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advantages from the standpoint of kinetics as to justify its adoption.

Experimental

The mineral was prepared from a stock of sphalerite from the Tri-State District supplied by the Eagle-Picher Mining Co. The mineral was crushed, ground, sized, and purified by treating with hydrogen peroxide. The latter step seems to remove organic coatings that might be present and leaves the mineral in the nonfloating condition that we expect the pure mineral to display. The washed, sized, and deslimed mineral was stored in glass vessels and formed into ion-exchange columns immediately before use. Fig. 1 shows the general experimental setup. The columns consisted of 12 to 15 g of mineral in a length of about 10 cm and a cross-section of somewhat less than 1 cm. Since the particles used were in the size range of approximately 48 to 325 mesh, a large number were present in any cross-section so that the behavior of the column was substantially the behavior expectable in any tank of commercial size. That is, the wall effect, if any, was of minor importance.

In using the column, one problem was the development of a method of feeding solution so that the rate of flow would be constant. This was obtained by use of a relatively high head and by inserting a mechanical resistance (clamp) in series to that offered to the passage of solution by the mineral column. During a test, the flow rate was checked repeatedly and adjustment was made by variation in the setting of the clamp.

The radioactive tracer was received from Oak Ridge and kept in the usual fashion and with proper precautions. Blending of the active silver with ordinary silver nitrate was made to suit the needs of the various experiments, and the combined solution was fed from a constant-head storage tank.

An idea of the silver content of the effluent solution was obtained by passing effluent solution through a concentric-shelled flow counter similar to that developed by Barnes and Salley for the evaluation of potassium.[9] Prior use of a flow counter copied from the design of Kettelle and Boyd[10] was not particularly successful because that form of counter cannot be cleaned readily of silver ions that become absorbed on the walls of the plastic container.

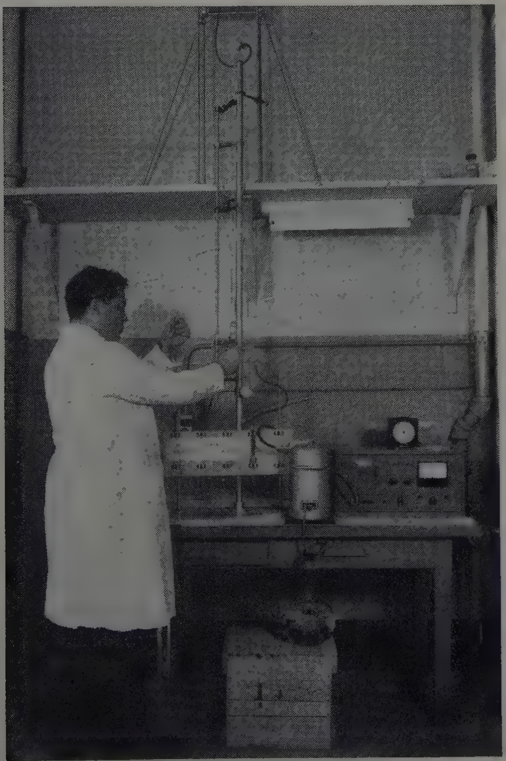

Fig. 1—General experimental setup.

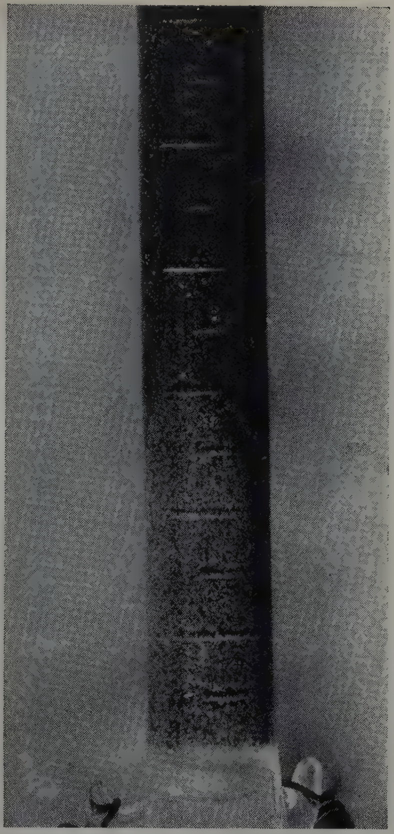

Fig. 2—Packed sphalerite bed showing discoloration progressing downward.

In addition, samples of the effluent were taken every few minutes for analysis. A standard volume of each sample was dried on a planchet and counted, using a standard end-window counter. After deduction of the background, use of the appropriate factor gave the silver analysis. The factor was such as to allow for the time-decay of silver 110, for the dilution of active silver with barren silver, for the position of the planchet below the window counter, and for the volume of solution taken for analysis. All these matters are now standard in the field of radio chemistry as may be noted by reference to one of the texts in that field.

In general, the indications given by the flow counter were at least quantitatively correct. Certainly the flow counter would indicate the incipience of break-through but the flow counter would not give accurate analyses after it had begun to absorb silver from the solution.

Results

Aside from preliminary experiments in which the flow rate and mineral volume were varied, the experiments were all made at a flow rate of about 10.8 ml per min. Since the amount of mineral in the columns was of the order of 12 to 15 g (volume 3 to 4 cc) and since the volume of voids is of the order of 3 ml, it appears that the retention time of the silver-bearing solution in contact with mineral was of the order of 15 to 20 sec.

Under these conditions when the concentration of the influent solution was very low (10^{-6} mols per liter), all of the silver was removed by the sphalerite. This experiment was continued for 27 hr, and there was no trace of silver in the effluent to the very end of the test. The feed solution, in this instance, contained a relatively high proportion of radiosilver so that a concentration of silver in the effluent of 10^{-9} mols per liter would have been detected readily.

This experiment shows that in spite of the short period of contact between a given volume of solution and the mineral column, all of the silver ions have an opportunity to reach the sphalerite, and that all of them take advantage of it.

If higher concentrations of silver are used in the influent solution, the abstraction is again complete at the beginning of the test. After a measurable length of time, the effluent solution begins to carry silver. This shows that although the same opportunity exists for the silver ion to reach the sphalerite, silver ion fails to stick after the mineral has acquired a sufficiently dense coat of silver. Of course, zinc ion was given up for the silver adsorbed. This was determined colorimetrically through the use of dithizone as indicator.[11]

The reaction with silver ion discolored the sphalerite which turned darker, see Fig. 2. In some experiments, in fact, the mineral became quite black. The initial material was the honey-colored mineral from the Tri-State District, which has been the common laboratory material in flotation experiments with sphalerite.

The course of the experiments is typified by Fig. 3, which shows the silver uptake as a function of the silver throughput. The figure consists of two parts. The upper part shows the instantaneous uptake and the lower part the cumulative uptake. The three curves for slightly different pH values (from 5.4 to 6.4) overlap almost exactly in the instantaneous curve, yet differ slightly in the cumulative chart. This is because the ordinate in the upper chart disregards the weight of mineral in the column while the ordinate of the lower chart allows for it. Fig. 3 shows that up to the break-through time denoted by point A, the amount of silver abstracted from the solution by the mineral is proportional to time; this amount, of course, being the entire quantity of silver coming to the column with the influent solution. Past the break-through point, the mineral is still removing silver from solution, but it fails to remove all of it. During the second period, the removal of silver is at a decreasing rate, as the coating of silver thickens.

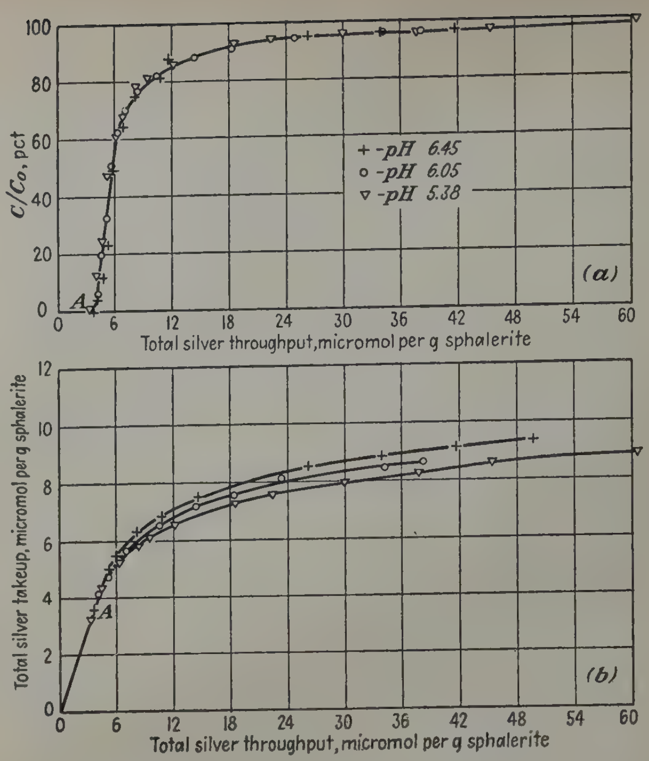

Fig. 3—Concentration histories showing effect of varying pH.

Clearly we are faced with the likelihood that two reactions are involved. The first seems to go to completion within the time allotted by our experimental procedure; it is our opinion that providing diffusion of silver ions within the percolating liquor can take place sufficiently to bring every silver ion to the surface, an opportunity for completion of the first reaction will have been given. The second reaction does not appear to depend on any factor arising within the aqueous solution.

Further evidence of the existence of this second phenomenon was obtained quite fortuitously during one of the experiments. The experiment involved the use of a rather dilute influent solution so that when the day was over, the experiment had not been completed. The flow of solution was stopped overnight and was resumed the next day without disturbing the column. When the flow was interrupted, the column was still abstracting some silver but not all of it. On the next day, the mineral was found to be much more avid for silver, hungry

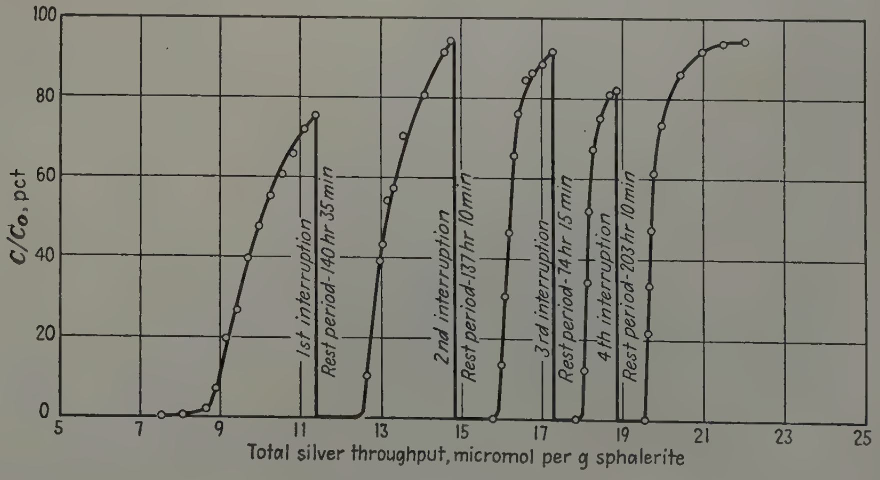

Fig. 4—Concentration history showing effect of interruptions.

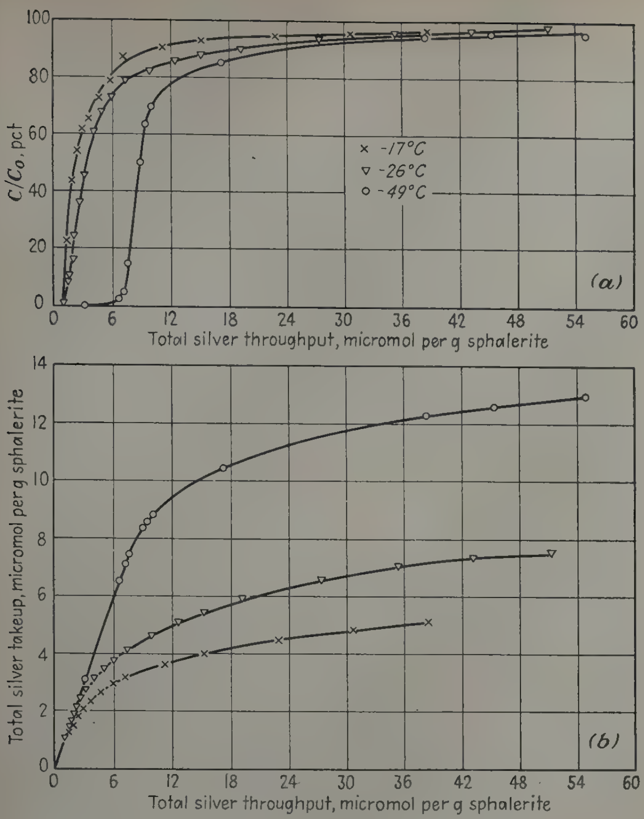

Fig. 5—Concentration histories showing effect of varying temperature.

enough, in fact, for it to completely deplete the influent solution. This was so intriguing that the main purpose of the experiment was changed. The flow of solution was continued throughout the remainder of the week with nightly interruptions. Apparently here was an instance where in a sense the mineral got tired of picking up silver in the usual course of ion exchange but where the activity of the mineral was restored by rest.

The interpretation of this phenomenon is that there is a diffusional rearrangement within the solid going on all the time. During the flow of solution, the rate at which this rearrangement occurs is so slow that few new loci are provided for the adsorption of silver. But when the mineral is at rest, the rearrangement is sufficiently extensive to provide considerable adsorption capacity—in fact enough to entirely deplete the liquor in the time of its passage through the column. The system behaved in the same way at each cycle. The experiment was again repeated with longer interruptions, as may be seen in Fig. 4.

On wholly speculative ground, it may be supposed that the silver ions and perhaps the zinc ions also diffuse so as to form localized polymolecular crusts alternating with barren spots. Such an effect could occur if proper values are assumed by the various interfacial energies.

On the other hand, it may be assumed instead that the silver ion diffuses into the zinc sulphide lattice. The authors do not favor this alternative because it has been shown that both sulphur and zinc ions are not prone to diffuse in sphalerite,[12] but this alternative cannot be ruled out.

If either solid-diffusion hypothesis is correct, temperature may be expected to play a large part in the control of the second reaction, i.e., the slow reaction. That this is the case may be seen from Fig. 5, which presents the results obtained at different temperatures: at a high temperature, the slow reaction is much more rapid than at a low temperature. This suggests that at the temperature of boiling water the break in the cumulative curves may not be noticeable, while in freezing water, the slow reaction may be markedly slower.

Within a limited range pH does not play a major role in the action of silver ion on sphalerite, see Fig. 3.

Solutions that differ markedly in silver ion concentration, also, behave in much the same way. However, as a much greater volume of a dilute solution is required to carry the same amount of silver as a concentrated solution, the use of dilute solutions provides more opportunity and time for diffusional rearrangement. Fig. 6 shows that the rate of uptake at the end of each experiment corresponds to the formation of surfaces that still have some avidity for silver. The overall uptake is distinctly a function of the concentration, being more in the case of the more dilute solutions, as may be expected on kinetic grounds from the general character of slow-diffusional phenomena.

To provide the beginning of a comparison with the usual sphalerite activator, one experiment was made in which radioactive copper was substituted for radioactive silver. Also, another experiment was made with silver ion and copper-preactivated sphalerite. The results obtained are given in Fig. 7. It will be seen that in the copper-exchanging experiment there was a build-up in the adsorbed copper. In this case, however, the mineral took up a maximal amount of copper with no slow reaction to supplement the formation of the primary coat. In other words, copper behaves like silver at low temperatures, only more so.

Interpretation of the experiment on the copper-preactivated silver-exchanging experiment is not

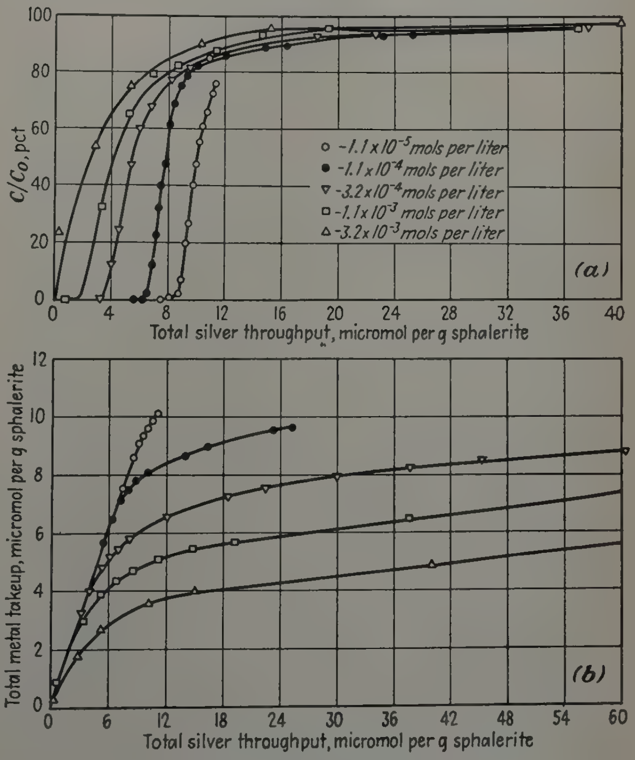

Fig. 6—Concentration histories showing effect of varying solution concentrations.

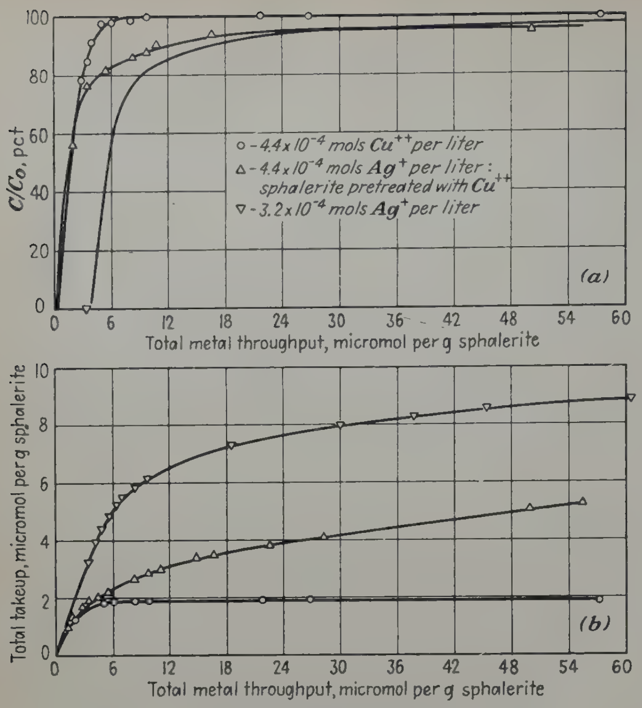

Fig. 7—Concentration histories showing comparison of copper with silver tests.

completely possible as it is not known whether Ag^+ exchanged for Cu^{++} or for Zn^{++}. Since the total Ag^+ uptake exceeds the total Cu^{++} uptake in the other experiment, and also on the chemical grounds, the Ag^+ is more likely to have exchanged with Zn^{++}. Tracer experiments in this direction should be profitable.

The amount of copper picked up by sphalerite in the copper-exchanging experiment is not nearly as much as the amount of silver that can be picked up. Furthermore, a ceiling appears to exist.

Fig. 7b shows that the mineral will adsorb up to 1.89 micromols of copper per gram of mineral. On the basis of preliminary surface measurements by the gas adsorption method,[13-16] this uptake appears to come close to a complete, closely packed monolayer of copper ions. It seems, therefore, as if a determination of copper-ion adsorption may permit evaluation of the amount of surface on sphalerite. Further work is needed in this field before a conclusion is accepted.

It is obvious that further experiments with copper would be profitable and that similar experiments might be conducted with a number of other metals replacing zinc at the surface of sphalerite. Still another possibility involves a study of the joint action of metallic ions like silver and copper. It is entirely possible that one of these elements acts as a catalyst to facilitate the formation of a coating by the other. Such phenomena may be important in ore genesis as well as in ore-treatment processes.

Summary

A preliminary study was made of the adsorption of silver ion by granular sphalerite. The mineral was arranged as an ion-exchanging column, and the dilute influent electrolyte was marked with radioactive silver. At suitably low flow rates the solution is completely stripped of silver until the coating becomes substantial. The rapid ion exchange that occurs during flow of solution through the column is supplemented by a second or slow reaction which appears to be controlled by diffusion in the solid state.

In a single experiment with copper instead of silver at room temperature, no slow or solid-diffusion reaction was observed. Much additional work remains to be done.

Acknowledgments

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A Graphic Statistical History of the Joplin or Tri-State Lead-Zinc District

by John S. Brown

IN 1925 the writer undertook a detailed statistical study of all producing areas in the Joplin district as a basis for evaluating programs and measuring objectives. For this purpose, the published figures in the yearly volumes of Mineral Resources were used, supplemented for earlier years by publications of the Missouri Geological Survey and other local and less official sources. When all else failed, the available data were projected backward to hazard a reasonable guess as to the unrecorded early output of important areas. Fortunately, the proportion of such prehistory production is not a large factor in any of the totals. These results were used during the next few years to measure the relative importance of various producing areas and to predict the peak period of development of the all-important Picher field. For the purpose of this review, the charts have been completed to the end of 1950. During World War II, the U. S. Bureau of Mines became interested in a similar study and issued comprehensive statistical tabulations of data up to 1945 (Info. Circular 7383), which have been checked against the figures used herein. This tabulation, however, does not include all the earlier data used by the writer nor does it offer any estimates of the wholly unrecorded era in the beginnings of the earlier camps.

The area covered in this study is shown in Fig. 1 on which are indicated the relative location and approximate outlines of the principal producing camps. This also shows the approximate yield to date of each major camp in terms of combined lead and zinc concentrates. The output of zinc concentrates is roughly seven times that of lead. Hence, the economy of the district has depended primarily on the price of zinc, with lead as an important byproduct. Over much of the productive period, lead concentrates averaged about twice the value of zinc concentrates per ton, and in certain mines or areas the proportion of lead to zinc was substantially above average.

The Joplin district is largely flat prairie but is partly moderately dissected, partially wooded land with a relief generally less than 100 ft. The rocks are almost flat-lying, nearly parallel to the surface, and the chief ore formation is the Mississippian Boone limestone, including its cherty phases. This formation either outcrops in the producing areas or is covered by a thin veneer of Pennsylvanian shales. Virtually all the ore occurs within 400 ft of the surface, and a large part at less than 300 ft in depth. Most of the land was divided into small farms or town lots before mineral development; tracts seldom exceeded 160 acres, and averaged considerably less. Mineral rights followed the surface ownership, segregation was rare, and a system of leasing for mineral development became well established early in the region's history, many landowners deriving small to sizable fortunes from royalties. Because of the shallowness of the ore and other factors, prospecting and mining was cheaper than in almost any comparable mining district in the United States. This situation, coupled with the widely divided land ownership, offered a fertile field for promoters and speculators and led to the rise of many small mining concerns. Only in its later history, under stern economic compulsion, has control tended to centralize in a few companies.

Under these conditions, any important new discovery or successful development had much the effect of a gold rush or an oil boom. Every property in the area was leased quickly, promptly drilled, and, if ore was found, it was soon on the market. Many companies and individuals participated, and the average producing lease-hold probably was about 40 acres in extent. Any important field thus was attacked by anywhere from 10 to 100 or more producers. Production zoomed, eventually steadied or wavered, and ultimately subsided, leaving a desolation of tailings mountains, cave-ins, empty housing, and wreckage. The object of this paper is to depict the pattern of this process, so far as metal production is concerned, and to note the way in which it reacted to economic and political pressures.

Production Charts

In Fig. 2 is charted the production record, in tons of lead and zinc concentrates combined, of eight of the principal camps, which together account for approximately 99 pct of the total district production, over the years from 1870 to 1950. This period covers all but the very minor beginning of mining history. Two important camps are divided by state lines; hence, it has been necessary to combine production records for the two portions, based on estimates that may be slightly in error. Certain camps are subdividable into important units for which separate figures are available in whole or in part and have been charted as fractions of the major unit. The corresponding price of zinc is shown above all the charts.

Three camps, Aurora, Neck City, and Galena, show a remarkably symmetrical graphic pattern, which is interpreted as the norm. The curves rise steeply to a peak, level off for an irregular interval, and then drop sharply to zero on a slope corresponding roughly to that covered by the initial rise. The three portions of these charts seem appropriately characterized by the designations of youth, maturity, and decline. On the whole, with some irregularities, the production in each of the three periods seems to be almost equal.

A fourth camp, Granby, fails to conform to the normal pattern. It exhibits a very long period of reasonably uniform, stabilized production corresponding to maturity, followed by a rather precipitate decline. Its youth is hidden in the era of prehistory. This habit of steady, long-continued production at an even keel is attributable to the fact that this camp, more than any other, was controlled largely by a single principal owner at any given period over most of its history and this permitted the imposition

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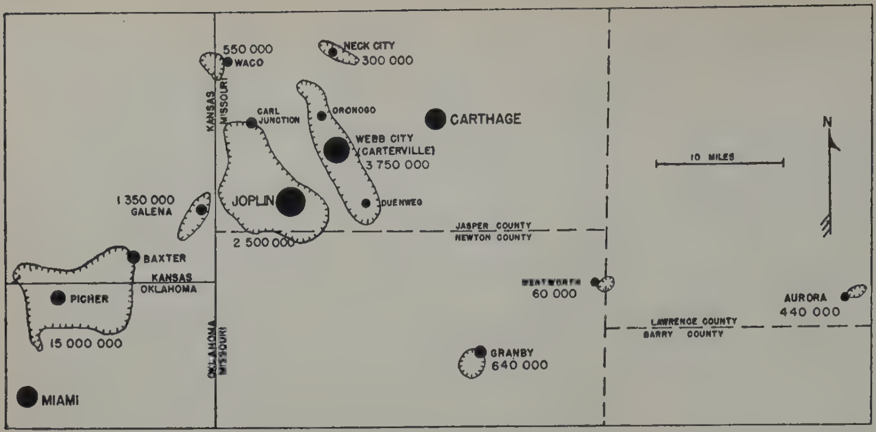

Fig. 1—Index map of Joplin district.

of a much more orderly plan of development and production. Oronogo, a subdivision of Webb City, shows a rather similar curve for much the same underlying reason. No doubt this pattern was much more satisfactory for the operating companies, probably was better for the communities, and likely resulted in the recovery of a greater percentage of the metallic wealth.

The greatest of all the camps, the Picher district, first began operations in its most southerly corner about 1908. It has since produced considerably more than all the other camps combined and shows the next most regular pattern. Failure to appreciate the significance of the new camp, coupled with a serious water problem, caused it to develop slowly for a few years. However, its importance became apparent about 1915, and it rapidly ran the normal cycle, reaching the peak of youth and entering maturity in 1926. The mature stage was interrupted from 1930 to 1935 by the Depression, although a substantial production was maintained even in the hardest year of this adverse period. Maturity apparently terminated in 1941 (or possibly 1937) and the field is now well down the path of decline. The chief abnormality of this curve was caused by the Depression. The total yield in maturity was moderately greater than that of youth, and the period of decline, which now shows one third to one half the yield of maturity or youth, would appear to be about half completed.

A somewhat similar but more abnormal history is that of the Waco camp. Youth, 1916 to 1923, had the usual steep rise. Maturity ran into the Depression, which brought complete cessation of production in 1932, with an irregular resumption in the late thirties and the ensuing decade. No normal segregation of maturity and decline seems feasible, but the end of both appears to be now at hand. The yield of the combined periods of maturity and decline totals almost exactly twice that of youth.

There remain two important and unusually complicated situations represented by the Joplin and Webb City fields in Missouri. These together, and roughly in equal proportion, were the mainstay of production in the period from 1900 to 1918, supplying more than two thirds of the output of the district, each field achieving its cycle of maturity in this era. Suddenly, in 1917, production in both fields began declining and practically reached zero in 1920. This extraordinary behavior was the direct result of the sudden development of the much larger and richer Picher field. When the magnitude of this new discovery became apparent, there was a veritable stampede in which drills, mills, and personnel were moved to the new field. The problem of keeping the old fields dewatered under diminished output became insupportable, and the mines were abandoned and the workings flooded. In spite of the abandonment of these major camps, however, production from the new field flooded the market, and the American Zinc Institute was formed to promote new uses for the excess zinc.

The ready availability of so much equipment, manpower, and organization, transferrable from the old camps to the new, doubtless steepened the curve of development in the youth of the Picher field, and in 8 years, from 1918 to 1925 inclusive, the inevitable peak of youth was attained. For a field of equal size in a wholly new or more remote area this growth likely would have taken more than twice the time. On a minor scale, a similar history characterized the Waco field.

The brief postwar depression of 1921 caused a slight hesitation in the youthful progress of Picher and Waco. The depression of the thirties, however, put Waco out of business completely and, but for the presence of a few stronger and better integrated companies which had now developed, might well have done the same for Picher.

The Price Factor

Thus far one factor in these charts has been ignored, which often has been considered the prime influence on production. This is the element of price. So far as these individual camps are concerned, the graphs seem to indicate clearly that relative age and productive capacity, which is closely related to grade of ore, are far more important factors than price, the effect of which is too greatly modified by being spread over the entire industry. Picher and Waco achieved the first half of youth at a prodigiously steep rate in the face of prices that were declining drastically. Aurora completed the latter half of youth on a declining price. Galena, Granby, Joplin and Webb City show moderate fluctuations in youth, which probably reflect vaguely the price fluctuations of the 90's, without, however, seriously modifying their general upward trend. In only one of the eight major camps, namely Picher, is the peak of youth coincident with that of a sustained price rise, which occurred in the latter half of youth, and here the peak of production is out of all proportion to the moderate increase in price.

The end of maturity probably is somewhat more affected by price fluctuations and economic stresses. For Aurora and Galena, it seems to be related fairly closely to the peak of a sustained price rise. Neck City and Joplin ended maturity almost coincidently with the long and gradual rise of prices prior to World War I, and the ensuing war boom failed to modify the subsequent downward trend, although, for Joplin, it nearly succeeded in extending the mature stage. Webb City, and perhaps Granby, developed abnormal peaks marking the apparent end of maturity with only a slight delay in relation to the fantastic price rise of World War I. Waco was too much affected by the Depression for a clear interpretation. Picher ended maturity about in the middle of a long-extended rise in prices related to World War II, and neither premium prices during the war nor inflated markets thereafter served to modify its downward trend.

Both maturity and decline show moderate correlation between volume of production and swings in the price level, and the several periods of decline probably would be tied closely to this factor if it were not for more violent external effects, such as wars, depressions, and the discovery of the Picher field. Three camps, Aurora, Neck City, and Galena, were closed by the price deflation following World War I, but, since this happened also to coincide with the great expansion of Picher, the cases are not clear-cut. Joplin and Webb City had their periods of decline amputated at the beginning by the same fate, and Granby was in much the same situation. The decline of Picher and Waco undoubtedly has been retarded in the past 5 years by the current price inflation.

The Grade Factor

A factor equally or more important than the price of metal, apparently, is the grade of ore available for production, although, unfortunately, the published statistics of grade are not complete enough for thorough analyses of the relations. Mineral Resources and the succeeding Minerals Yearbook give the percentage content in zinc and lead under the headings of the three areas, Southwestern Missouri, Kansas, and Oklahoma, from 1909 to the present. For Oklahoma, this grade may be assumed to be synonymous with that for the Oklahoma portion of the Picher field and for Kansas the contribution of Waco, Galena, etc. was so small (approximately 8 pct on the average) that the Kansas grade may be used to represent the Kansas side of the Picher field without serious error. On that basis the two have been combined in proportion to their respective tonnages to represent the grade of the Picher field, shown on the graph of production. It will be seen that the camp began operating on ore averaging about 5 pct combined zinc and lead (metallic content) and has declined unevenly but fairly consistently to a yield of exactly half, 2½ pct. (In all these grade figures the grade used is for crude ore ignoring that of remilled tailings. Concentrate production, however, includes tailings production).

The beginnings of the Missouri field are uncertain as to grade of ore prior to 1909 and thereafter are doubly complicated by the existence of numerous separate camps yielding a considerable variety of ores. In the two major camps, Joplin and Webb City, two distinct classes of ore were mined, "soft ground" and "sheet ground," respectively, the former more pockety and richer, the latter more uniform in occurrence but of lower grade. Grade and tonnage for the two classes were listed separately from 1909 to 1920, but they are combined here into a single grade weighted for the respective tonnages to give the grade line on the Joplin and Webb City production chart, Fig. 3. This shows that the average ore mined in 1910 contained 2 pct combined zinc and lead and the grade declined at a rate that would have reached 1½ pct in 20 years had not production ceased in 1920. Projecting this line backward, we may infer that the average grade of all ore mined prior to 1910 must have been between 2½ and 3 pct metal content, and in the early years of richer camps probably reached 4 to 5 pct. At the time Picher became important, however, the available grade of ore in the new field was substantially more than twice that then available on a tonnage basis in Missouri. Naturally, only a rich mine with a reasonable water situation could survive outside the Picher field.

With the closing of Joplin and Webb City in 1920, the only important remaining production in Missouri was that of the Missouri side of the new soft-ground Waco camp. The Missouri grade figures from 1920 on are plotted on the Waco production chart. In the important decade from 1920 to 1930 they probably represent rather accurately the actual grade of the Waco field, even including its Kansas portion. Thereafter, the grade is influenced appreciably by revivals of Joplin, Webb City, etc. but still probably is not too far from the true Waco line. The initial grade at Waco seems to have averaged about 3½ pct falling to a present and final level around 2 pct.

If production and price fail to show any close interrelationship, the same is by no means true of price and grade. Compare the Picher grade curve with the price line and, except for the long-range downward trend of the grade line, all fluctuations in the two are virtually mirror reflections of each other; when the price went up, the grade of ore mined dropped and vice versa. The same is true of the Missouri curves. This is merely the operators' means of adjusting the fluctuations in value of his selling product (concentrates) to meet the relatively more stable elements of cost and leave an operating margin. When concentrates yield less money, he mines his richer ore to obtain more concentrates per ton of ore at a reduced cost; when metal prices rise, he mines the lower grade ore, which formerly failed to yield a profit. This procedure is satisfactory within its limitations but cannot alter the downward trend of the grade line over the life cycle of a district or camp.

The question may be raised as to whether or not the declining grade curve may represent largely technological progress—the lowering of costs and improvement of methods to make ore of what formerly was waste, and whether or not this process might not continue almost indefinitely. In the author's opinion, there is a small fraction of truth in this idea. There is, of course, some low grade marginal ore in this district. But in the case of all soft-ground camps, including Picher, which is more closely allied to the soft-ground deposits than to the sheet ground, it is believed to be less in total volume than the original ore mined and far less in potential yield of concentrates. In the Webb City and Joplin camps, where sheet ground, a low grade form of mineralization widely distributed in rather gradational habit, was an important contributor to the output, the idea has some validity but probably is exaggerated easily.

Probable Reserves

If the foregoing review and analysis have any value other than historical it should be in: 1—assisting in the evaluation of any new camps that may be discovered, and 2—aiding in a forecast of the possible remaining potential of existing camps. Only the second item will be discussed here.

First, it seems clear that the greatest potential in the district still is that represented by the declining phase of the Picher field. Youth yielded something over 5 million tons, and maturity, as charted, over 7 million tons of concentrates. The peak year (1925) was almost equalled by the succeeding year (1926), and if the latter were used as the dividing line, the two periods would be almost equal at a little over 6 million tons each. Accepting this interpretation as a tentative yardstick, 6 million tons may be expected on the decline, of which 2,244,000 have been produced, leaving 3,756,000 expectable. Considering the recent difficulties of maintaining current output, even with insistently rising prices, this forecast seems optimistic. A second look suggests that the end of maturity could have been in 1937 (the Kansas side shows this clearly) except for the strong armament boom of 1940-41. Cutting maturity back to 1937 would reduce the production in that period to 5,680,000 tons, nearly equal to that of youth (1925 basis) and would augment decline to 3,986,000 tons, leaving a probable potential of about 1,750,000 tons of concentrates. The writer believes that the final production may be nearer the latter figure than the former, especially considering that declining fields are particularly vulnerable to stress and strain and that, once the heavy pumping cost for the field becomes unsupportable, production is likely to cease abruptly, unless artificially supported.

The indicated grade of ore to produce these concentrates, on the basis of the smaller yield, would be about 2¼ pct combined metal content, or about 80 million tons of crude ore, producible in 12 to 15 years. The larger figure (3,750,000 tons concentrates) would imply a life of 40 to 50 years with small yearly output in the latter half of the period, an average grade less than 2 pct, and a total tonnage of 175 million. This compares with a total crude ore output to date of a little over 400 million tons from this camp.

It is clear, likewise, that the Webb City and Joplin fields must have a substantial reserve of ore, *which would have been produced had not Picher been discovered.* (Whether or not this ore is still producible from this reserve is another matter). For Webb City, the charts show a production of a little over 2 million tons of concentrates in youth, a little over 1 million in maturity, and nearly 600,000 in the decline, including in this the revival of 1935 to 1950, which has netted 300,000 tons. Averaging youth and maturity at approximately 1,620,000 each, and assuming decline to have equal value, it would seem that 1 million tons of concentrates may remain in this field. *The grade of ore to yield this, however, will not much exceed 1½ pct at the start, and would almost certainly drop to 1 pct or below at the end.* A million tons from 1¼ pct ore, or 80,000,000 rock tons is indeed a challenge to the champions of improved methods and lowered costs. They may retort that this is but little lower than the grade mined in the era prior to 1915. As discussed later, however, that was another era.

The Joplin camp, with 1 million tons of production in youth, 1 million in maturity, and ½ million in decline, suggests the probability of exactly 50 pct of the potential of Webb City; ½ million tons of concentrates from 40 million tons of rock at 1¼ pct grade.

The remaining camps seem to offer only small possibilities for additional production. However, a careful appraisal of Neck City might be rewarding. The period of decline for this camp, although superficially normal, was short and steep, the yield less than 50 pct that of youth, less than 33 pct of maturity. A plausible explanation for the rapid decline would seem to be that the rush to Picher and Waco caused its abandonment before its exhaustion. Perhaps 50,000 to 75,000 tons of concentrates exist here, and the grade of the crude ore may be a bit above that of Webb City and Joplin. However, the water problem is sizable.

It may be noted that World War II stimulated great efforts in the Tri-State district to maintain or expand production, both on the part of private operators and governmental agencies. The premium price plan was devised probably more with a view to increasing production in this area than for any other single objective. Under it there was substantial exploration; numerous camps were re-examined, prospected and revived, the results, in the main, are shown in Fig. 2. Moderate production was obtained from several dormant camps and moderate increases from others. As a commentary on this program, there is inserted the historical chart of Wentworth, a minor camp but the only one which yielded, under this program, any large production in comparison to its past history, in this case some 30,000 tons of concentrates, approximately equal to its past production. The future of the Joplin district seems to be, if anywhere, in an extensive long-range program of searching for new major camps or entirely new producing horizons in the existing camps, with no guarantee of success.

The Price-Grade Multiple

The implications of the price-grade relation seem so significant as to deserve special consideration, particularly with reference to the matter of possible reserves. If one reacts inversely to the other, it is obvious that by multiplying them something resembling a constant may be obtained, and this might be a useful figure in estimating the price necessary to yield production at a given, and lowered, grade of ore. This idea is illustrated in Fig. 3, which gives the price of zinc from 1900 to 1950, in comparison with the grade of ore from the Missouri fields on one hand and Picher on the other. It also charts the multiple of price for these two areas.

The resultants are by no means smooth curves but they are considerably less erratic than either price or grade. Noting first the Missouri price-grade curve, it plainly shows a division into two eras, one prior to 1915, for which grade figures are available only to 1909, but which, it may be inferred from general knowledge as to grade and by an inspection of the price chart, would undoubtedly hold good back for many years, probably to 1880, or 35 years. This was the era when mine wages ranged from perhaps as low as $1 to a maximum of about $2.50 per day, with no social security, no unemployment insurance, little health or safety provision, and low taxes. In that era the price of zinc averaged about 5¢ per lb with only mild fluctuations compared to the succeeding period and the price-grade multiple ranged between 10 and 15, averaging a little less than 12 in the later years.

The inflation of World War I ushered in a new era which has likewise run 35 years to the present day. The price of zinc, although fluctuating more widely, has averaged about 8¢ per lb for these 35 years, an inflation of 60 pct. Coincident with this inflation came the rise of the Picher field with its richer ore, and the average grade of ore mined in the district as a whole approximately doubled. Inevitably, as a

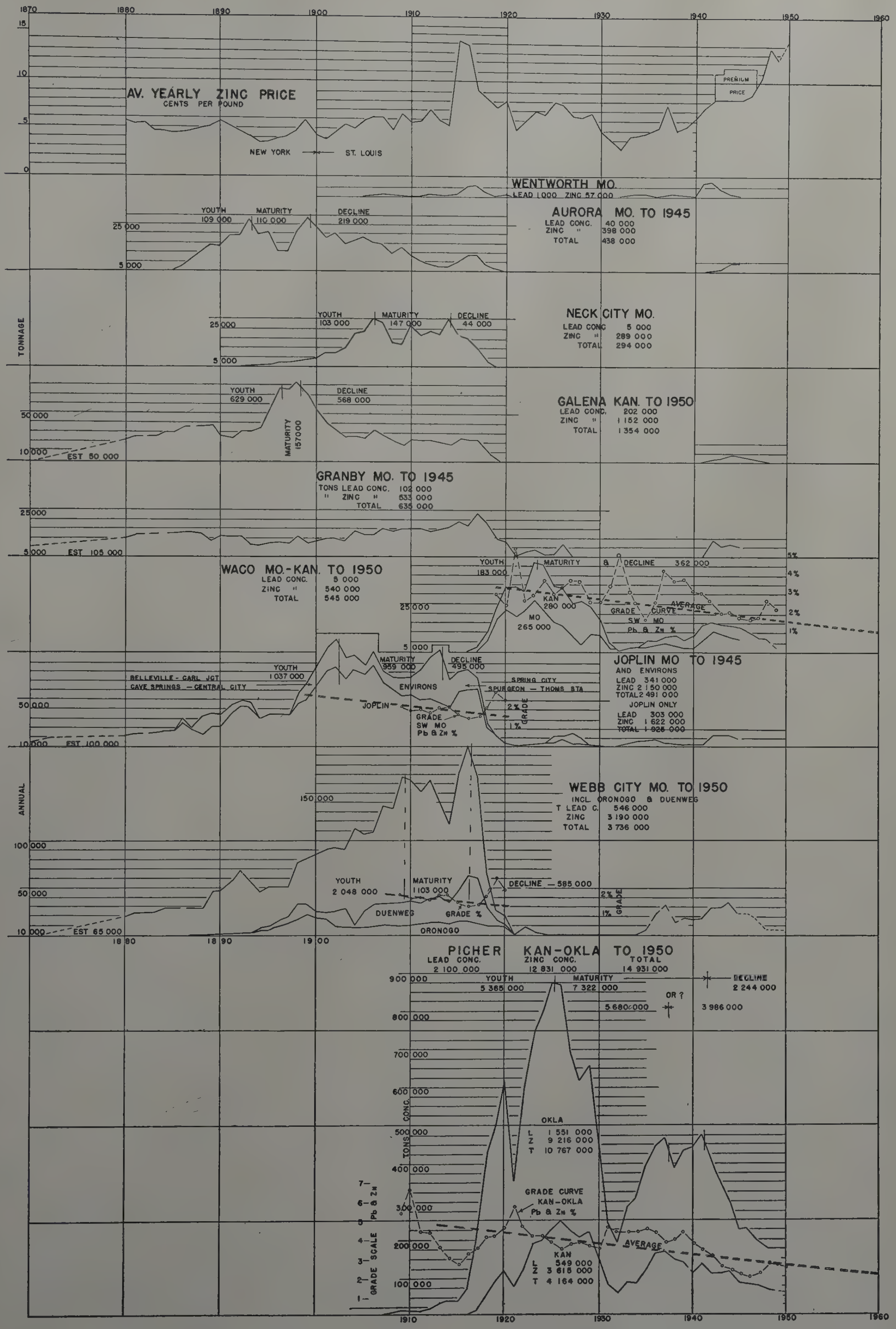

Fig. 2—Production record of the principal mining camps in the Joplin district.

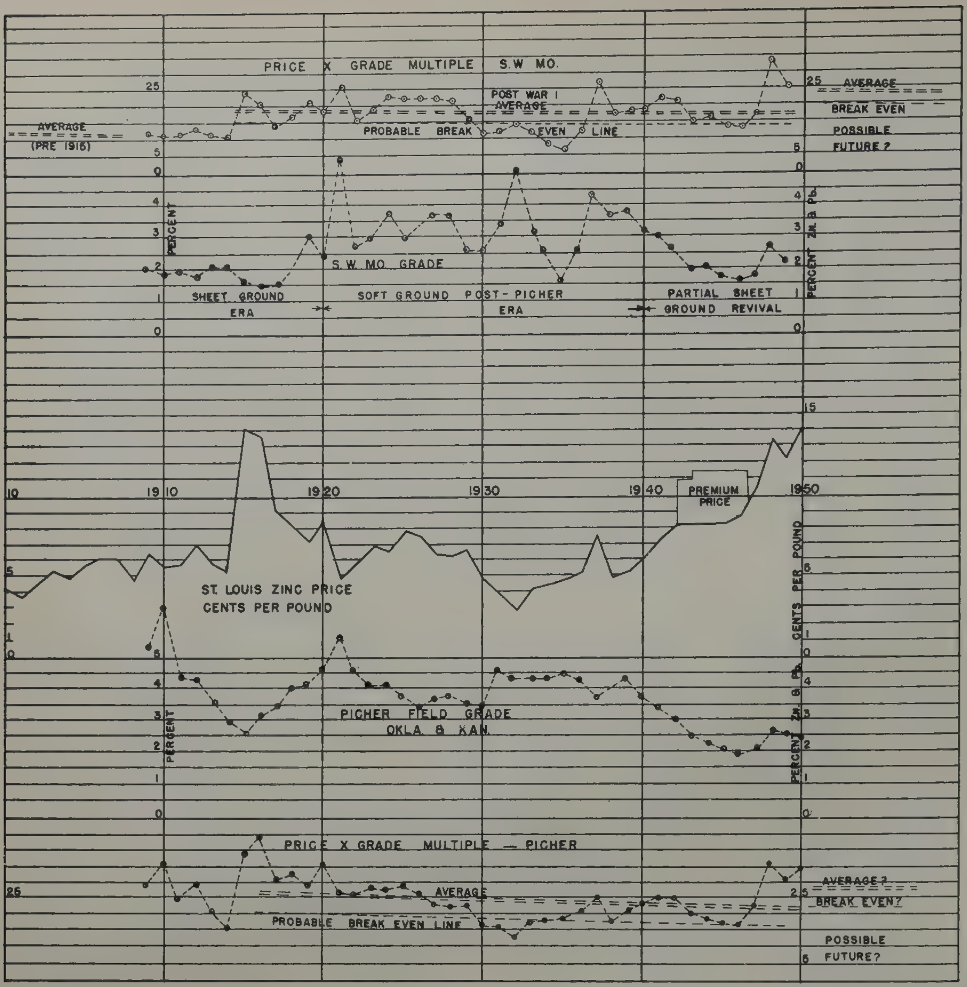

Fig. 3—Price-grade relationships in the Tri-State district.

result of these two controlling elements, the price-grade multiple rose to a higher average level, practically double that of the former era. For the Picher field, it is approximately 25; for Missouri, after 1915, about 18. In all cases, there is a slight declining tendency from the beginning to the end of an era, perhaps explainable as caused by technological improvement in part (lowering of costs and improvement of processes to permit mining lower grade ore) and perhaps in part to the liquidation of the capital charges (for Picher) permitting operation at lower cost and holding down prices to some slight extent.

Prediction is hazardous, but the suggestion seems warranted that the sharp and insistent rise of the graphs for the multiple in the past 3 years (1947-50) presages the beginning of a third era, perhaps another 35 years, more or less. The writer suspects that the price-grade multiple is leaping to a new level and will never return to its former position.

Assuming that the multiple, which now stands at 35 for the Picher field, may average 30 over the next decade, then the requisite for mining 2 pct ore is a 15¢ zinc price; for a 1½ pct ore, 20¢. This is not to imply that production would cease below 15¢, for the probable break-even line is somewhat lower, and dips below the 15¢ price could be weathered for moderate periods.

In the Missouri field, where it appears the average of the multiple may rise to only 25 on a basis corresponding to Picher, the requirement for mining 1½ pct ore would be about 17¢ zinc and for 1 pct ore a price of 25¢ is indicated. Since the Tri-State area, owing to its decline in output with a corresponding increase from other sources, has lost completely its power to dominate the market, it seems obvious that the price will be dictated mainly by outside influences. Whereas it would seem quite reasonable for the price to meet the requirements of production from the Picher field for a substantial period, it also seems unlikely that it will fulfill the requirements for the lower grade portion of the Missouri fields (the 1 to 1¼ pct ore). This issue is likely to be decided largely on a political basis. In doing so, however, the question of manpower may deserve careful scruitny. It is likely that any production from the lower grade portion of the Missouri reserve will require manpower per unit of product greatly in excess of that necessary in other areas. This might be a serious objection in some situations, and less in others.

Conclusion

The method of analysis offered here appears to have definite uses and limitations. It probably can not be applied at all, or only with drastic modifications, to simple veins or steeply dipping deposits where the area open to attack is limited in comparison with the extent of the deposits. It might fit well the situation of placer camps, residual deposits, and shallow zones of secondary enrichment. It should have considerable application to other deposits of the Mississippi Valley type, but each case doubtless will involve its own peculiar problems of adaptation to local geological, economic and historical influences.

Flotation Rates and Flotation Efficiency

by Nathaniel Arbiter

THE separation of minerals by flotation can be regarded as a rate process, with the extraction of any one mineral determined by its flotation rate, and the grade of concentrate by the relative rates for all the minerals. So regarded, the significant variables for the process are those that control the rates. These variables are of two types, the first describing the ore and its physical and chemical treatment prior to flotation and the second characterizing the separation process in the cells.

This paper will examine the variation in rates for a group of separations, will show that a simple rate law appears to govern, and will consider the relation of the control variables to the rates. The use of rate constants for evaluation of performance and efficiency will be discussed.

Flotation involves the selective levitation of mineral and its transfer from cell to launder. The flotation rate is the rate of this transfer. It may be defined by the slope of a recovery-time curve for any cell in a bank, or at any time in batch operation. The objective in flotation rate study is an equation expressing the rate in terms of some measurable property of the pulp. This can be either the concentration of floatable mineral in weight per unit volume[1, 2] or a relative concentration, which will be a function of the recovery.[3] A rate equation for an actual flotation pulp will contain at least two constants, both to be determined from the data. One of these, the initial concentration or proportion of floatable mineral, is not necessarily equal to the feed assay because of nonfloatable oversize or locked particles.[3] The other, a rate constant, is a measure of proportionality between the rate and the pulp property on which the rate depends. The value of the rate constant will be determined by the values of all variables which control the process and will be changed by significant changes in any of them. It is, therefore, a direct measure of performance. Where recovery or grade change continuously with flotation time, the rate constant will be independent of time and will characterize the entire course of the separation.

Development of Rate Equations

Rate equations can be developed either by analysis of the mechanism of the process or by direct fitting of equations to recovery-time data. Sutherland's attempt by the first method[1] suggests that the effect of particle size variation on the rate complicates the derivation of a simple equation applicable to an ore pulp. A further problem with an ore is the concentrate grade requirement, which usually involves a variable rate of froth removal. Thus the final rate for any cell may depend on the froth character and froth height, as well as on the pulp composition. This does not imply that each cell cannot reach a steady state[2] in which the rate will depend ultimately on pulp composition.

The second method is the fitting of rate equations consistent with the necessary boundary conditions* to experimental recovery-time curves. On the assumption that under constant operating conditions the flotation rate is proportional to the actual or relative concentration of floatable mineral in the pulp, a generalized rate equation may be expressed as follows:

$$\text{Rate} = Kc^n \qquad [1]$$

where K is the rate constant, c is some measure of the quantity of floatable mineral in the pulp at time t, and n is a positive number. In previous rate studies, the value of n has been taken as 1, either by direct assumption,[3] or as a result of the hypothesis that bubble-particle collision is rate determining.[1] A first order equation results, which after integration in terms of cumulative recovery R, leads to

$$\text{Log}_e \frac{A}{A-R} = Kt \qquad [2]$$

The quantity A is the maximum possible recovery with prolonged time under the conditions used. No conclusive proof for the validity of this equation in flotation has been advanced. The evidence cited in its support consists entirely in the demonstration that it appears to apply to a limited number of recovery-time curves.[3-5] It will be shown subsequently that this procedure is not sufficient to establish the order of a flotation rate equation. The possibility that the equation may be of higher order therefore requires examination.

If, in particular, the exponent in eq 1 is assumed to be 2, then after integration there results

$$R = \frac{A^2Kt}{1 + AKt} \qquad [3]$$

with K again a rate constant and A the maximum proportion of recoverable mineral. Eq 3 may be

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* At zero time recovery is zero; as time becomes large, recovery reaches a maximum.

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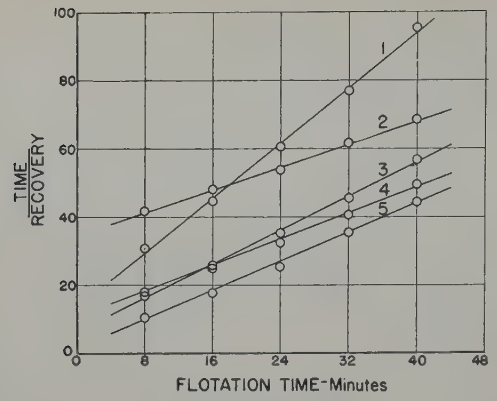

Fig. 1—Graphical test of second order rate equation. Data of Garcia Zuniga.

1—Coniapo gold ore **3—Elisa de Bordo silver ore**
2—Gold flotation tailing **4—Oxidized copper**
5—Sulphide copper

arranged as follows:

$$t/R = 1/A^2K + t/A \qquad [4]$$

If the second order equation is valid, a plot of (cumulative flotation time)/(cumulative recovery) against time should result in a straight line. It will be convenient to redefine a rate constant in terms of the recovery relative to the maximum possible recovery A. This constant then becomes A^2K** and will be the reciprocal of the intercept on a plot according to eq 4.

It will be shown that available data fit eq 4 with reasonable accuracy. However, it will also be shown that curve fitting, or the calculation of constants from the data, will not usually distinguish between first and second order equations when applied to flotation data. A more sensitive criterion is used which suggests the validity of the second order equation for the data available.†

Recovery Rate Data

This recovery rate data have been considered:

1—Zuniga's test results on five ores[3]
2—Beloglazov's test results on chalcopyrite and sphalerite[4]
3—Sutherland's test result on glass spheres[1]
4—Original test results as follows:
   a. Laboratory flotation of chalcocite ore *A*
   b. Laboratory flotation of chalcocite ore *B*
   c. Laboratory flotation of malachite-quartz mixtures
   d. Continuous flotation of sphalerite from massive sulphide ore
   e. Continuous flotation of chalcocite from low grade ore
   f. Continuous flotation of molybdenite from low grade concentrate

To obtain the laboratory data in this investigation, concentrate samples were collected through timed intervals, and the recoveries as functions of time were calculated from the individual concentrate assays and weights and the composite feed assay. For the continuous flotation of the sphalerite ore, tonnage and assay data were supplied which permitted calculation of extractions for each cell. For the continuous flotation of chalcocite, only time samples of the cell concentrates were available so that relative rather than absolute recovery values were used. For the molybdenite flotation, time samples of cell concentrates as well as samples of cell feed and final tailing were obtained. Cell recoveries were calculated from appropriate assays and weights. Cell residence times in the sphalerite flotation were supplied. For the chalcocite and molybdenite circuits the times are estimates from tonnage and volume data. The complete data are listed in Table I, with recoveries expressed in decimal form.

Testing Data

The data are plotted according to the criterion of eq 4 (t/R against t) in Figs. 1 through 9. These show a reasonably close agreement to the linearity required by the second order equation. *The method of plotting becomes insensitive at longer flotation times. Hence agreement of the earliest points with the straight line is a necessary condition for the graphical test to be significant.* For two of the continuous flotation results plotted in Figs. 7 and 8, several of the early points deviate considerably from the straight lines. In each case this was associated with a restricted rate of frothing because of conditioning or concentrate grade requirements.

The first order equation cannot be tested graphically in its integrated form. Algebraic tests, involving the calculation of constants from the data and reproduction of the experimental curve, show in some cases as good a fit as that obtained with eq 4, in other cases a poorer fit. This approach does not permit a decision between the two equations.

The ambiguity is caused by the fact that in most flotation circuits or batch tests a large proportion of the floatable mineral is recovered in the first cell of the bank, or in the first flotation interval. The nature of the remaining portion of the recovery-time curve is such that several equations will approximate it.‡

In chemical kinetics, when such uncertainty exists because of side reactions or experimental inaccuracies, the order of a reaction is determined by the relationship between reaction rate and initial reagent concentration.[7] For a first order reaction, the fraction of material decomposed after any fixed time is independent of the initial reagent concen-

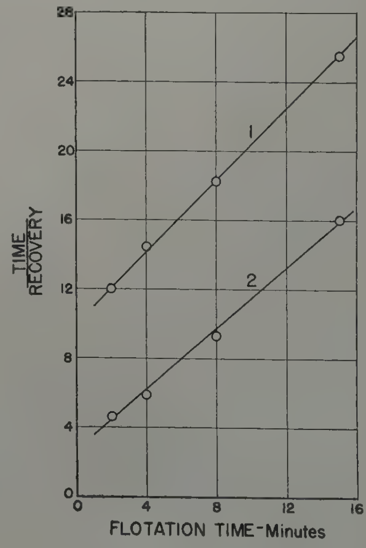

Fig. 2—Graphical test of second order rate equation. Data of Beloglazov.

1—Sphalerite
2—Chalcopyrite

** Rate = $A^2K(1-R/A)^2$

† Subsequent to the development and testing of the second order equation, it was found that Volkova[6] through an analytical approach had suggested that an equation similar in form should apply under certain conditions. Although he gave no experimental confirmation, Volkova's publication antedates the present investigation.

‡ An equation of the form $Rt = A + Bt$ will reproduce the major part of many recovery-time curves but is inconsistent with the condition that recovery must be zero at zero time.

Table I. Recovery-Time Data

Experiment Identity	Time, Min	Recovery				
Zuniga		Coniapo Gold Ore	Gold Tailing	Silver Ore	Oxide Copper	Sulphide Copper
	8	0.475	0.191	0.445	0.260	0.792
	16	0.627	0.333	0.652	0.358	0.895
	24	0.683	0.445	0.745	0.398	0.907
	32	0.702	0.521	0.792	0.416	0.910
	40	0.706	0.584	0.808	0.420	0.910
Beloglazov			Chalcopyrite	Sphalerite		
	2		0.434	0.166		
	4		0.677	0.277		
	8		0.862	0.439		
	15		0.939	0.589		
Sutherland		Glass Spheres (52x72 Mesh)				
	0.5	0.388				
	1	0.714				
	2	0.821				
	3	0.870				
	4	0.896				
	5	0.910				
Copper A		Copper	Iron	Copper	Iron	
		(Lab Machine 1)		(Lab Machine 2)		
	1	0.632	0.353	0.551	0.237	
	2	0.756	0.433	0.711	0.335	
	3			0.773	0.381	
	4	0.815	0.496	0.800	0.406	
	6	0.831	0.528	0.822	0.435	
	8	0.840	0.551	0.832	0.453	
	10	0.850	0.573	0.840	0.475	
	12	0.860	0.592	0.846	0.493	
Copper B		Copper	Iron			
		(Lab Machine 1)				
	0.5	0.491	0.394			
	1	0.649	0.537			
	2	0.755	0.663			
	4	0.818	0.734			

Continuous Flotation	Sphalerite			Chalcocite — Low Grade Ore		Molybdenite from Low Grade Concentrate	
	Cell No.	Time, Min	Recovery	Cell No.[a]	Recovery[b]	Cell No.[c]	Recovery
	2	1.7	0.1475	1	0.477	1	0.381
	3	3.3	0.5298	2	0.488	2	0.471
	4	5.0	0.5953	3	0.578	3	0.574
	5	6.6	0.6300	4	0.848	4	0.649
	6	8.3	0.6505	5	0.900	5	0.684
	7	9.9	0.6607	6	0.940	6	0.721
	8	11.4	0.6659	7	0.966	7	0.741
	9	13.0	0.6710	8	0.980		
	10	14.4	0.6750	9	0.989		
	11	15.9	0.6776	10	0.994		
				11	0.997		
				12	1.000		

[a] Total flotation time 4 min.
[b] Relative.
[c] Total time 7 min; assumed equally distributed because of high ratio of concentration.

Table II. Effect of Feed Assay Variation on Recovery

Feed Assay, Pct Copper	1.11	0.56	0.28	0.21	0.12
Concentrate Assay, Pct Copper	21.67	10.35	9.85	3.51	2.81
Tailing Assay, Pct Copper	0.082	0.090	0.070	0.12	0.059
Ratio of Concentration	21.1	21.8	46.0	36.4	48.2
Extraction	92.9	84.7	75.8	45.2	50.2

[a] Feed assay variation obtained by preparation of quartz-malachite mixtures containing indicated copper percentages; pulp preparation and treatment were otherwise identical in all tests.

Table III. Tabulation of Rate Constants

	Ore or Mineral	Rate Constant		Ore or Mineral	Rate Constant
1[a]	Gold ore	0.08	4	Ore A, machine 1	
1	Gold tailing	0.03		Copper	2.5
1	Silver ore	0.2		Iron	0.7
1	Oxide copper	0.1	5	Ore A, machine 2	
1	Sulphide copper	1.7		Copper	2.0
2	Chalcopyrite	0.4		Iron	0.4
2	Sphalerite	0.1	6	Ore B, machine 1	
3	Glass spheres	2.5		Copper	2.1
				Iron	1.5
			7	Chalcocite	2.9
			8	Sphalerite	0.6
			9	Molybdenite	0.5
			9	Copper	0.008

[a] Refer to Figure of same number.

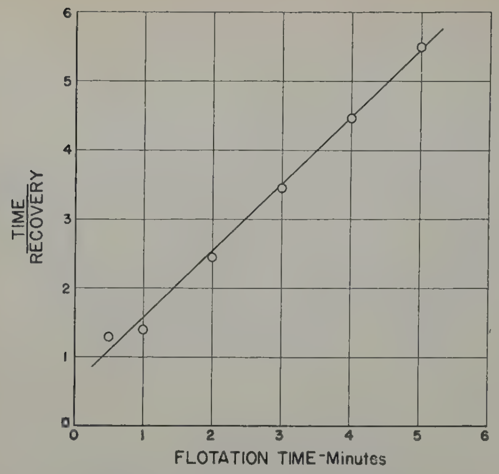

Fig. 3—Graphical test of second order rate equation. Sutherland's data on glass spheres.

tration. On the other hand, the fraction decomposed in a second order reaction will depend on the initial concentration. These same relationships apply to the flotation rate equations since they derive from the form of the equations and not from any assumptions about the mechanism of the process. The initial mineral concentration can be varied either by variation in feed assay or by variation in pulp density at constant feed assay.

Table II contains data illustrating the effect of feed assay variation on the recovery of copper from mixtures of malachite and quartz. The qualitative dependence of recovery on initial mineral concentration is indicated. Similar effects are described for feed assay variation with Climax molybdenum ore[8] for pulp density variation with galena-granite mixtures[9] and galena-quartz mixtures.[10]

This argument leads to the following conclusions:

1—A second order equation rate can be used to reproduce experimental flotation-recovery/time curves.

2—Rational support for a second order equation is supplied by instances of the dependence of recovery on initial mineral concentration. However, the limited amount of data of this type suggests that decision on the actual order of the rate equation be reserved.

For the purpose of this investigation, which deals with the use of rate constants in performance evaluation, the second order equation will be used.

Rate Studies Potentialities

The study of flotation rates can provide two types of information. First, knowledge of the order of the rate equation may lead to a better understanding of the mechanism of the process. Sutherland's attempt to establish that bubble-particle collision is rate determining, although not conclusive, indicates the approach. It may be possible through rate studies to determine whether such collision is involved, whether as Sun reports[11] multi-bubble aggregates can occur, and whether particle-particle collision can be rate-determining.[12] In addition, there is the possibility that the rate constant can be related to the contact angle, particle size, bubble size, and surface reagent concentration. Thus the rate approach may be able to integrate the fundamental flotation variables with those that measure the results of the separation.

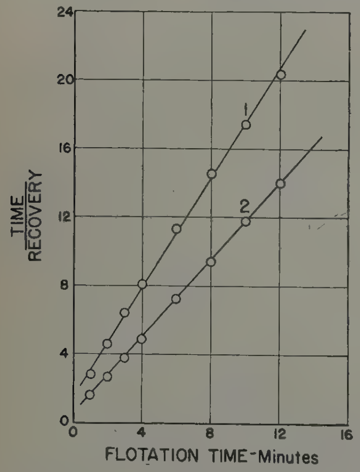

Fig. 4—Graphical test of second order rate equation. Copper ore A, laboratory flotation machine 1.

1—Iron
2—Copper

In the second place, rate studies can be of direct use in evaluating performance. The rate constant is particularly suited to this purpose. Table IV lists such constants for the recovery data in Table I. They were obtained either by extrapolation or by conventional algebraic methods.§ It has been indicated that the rate constant will be affected by changes within either of the two groups of variables controlling the process. To the extent that the data permit, the following discussion illustrates the effect of such changes.

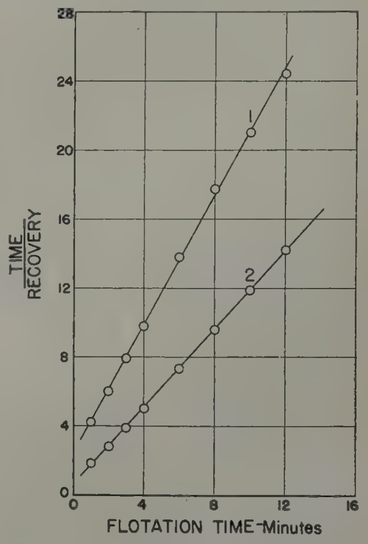

Fig. 5—Graphical test of second order rate equation. Copper ore A, laboratory flotation machine 2.

1—Iron
2—Copper

Rate Constant in Performance Evaluation

Variation in Ore and Ore Treatment: The rate constants listed in Table III range from 0.008 to 2.5, the variation being caused by variations in the minerals, in their treatment, and in the machines used. Differences between minerals may be caused by differences in those physical properties which affect particle size and shape in subsequent size reduction, or to a different response to reagent treatment. Zuniga's data show a wide variation in rate constant between a sulphide copper mineral in an ore and gold in a tailing, the former having a constant of 1.7 and the latter a constant of 0.03. Copper ores *A* and *B*, both containing chalcocite but from different operations, gave constants of 2.5 and 2.1 in the same machine and with similar reagent treatment. On the other hand, another sample of ore *A* gave a constant of 1.4, illustrating the significant differences in rate between samples from the same orebody.

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§ In terms of eqs 3 and 4, the relative rate constant A^2K is equal to $\frac{R_1R_2}{t_1\ t_2}\ \frac{(t_2-t_1)}{(R_2-R_1)}$, where R_1 and R_2 are the recovery values at t_1 and t_2.

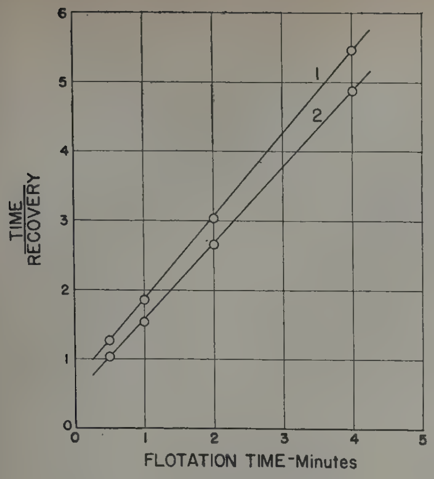

Fig. 6—Graphical test of second order rate equation. Copper ore B, laboratory flotation machine 1.

1—Iron
2—Copper

For two minerals floating from the same pulp the ratio of rate constants is a logical measure of selectivity.[4] It should be affected by any change in treatment which affects the rate for either mineral. The data in Table IV yield the following selectivity ratios, which are valid in each case only for the conditions used:

Chalcopyrite/sphalerite (Beloglazov)	4.0
Chalcopyrite/pyrite (Ore *A*, machine 1)	3.6
Chalcopyrite/pyrite (Ore *A*, machine 2)	5.0
Chalcopyrite/pyrite (Ore *B*, machine 1)	1.4
Molybdenite/copper	63

The selectivity ratio as defined above is closely related to Gaudin's Selectivity Index,[13] which was proposed without specific reference to relative flotation rates. For two minerals both of which are assumed to be completely floatable but at different rates, the ratio of rate constants at any time is $\frac{R_1}{1-R_1} \times \frac{1-R_2}{R_2}$, while Gaudin's Index is the square root of this ratio.

Variation in Machines: The selection of flotation machines for any operation is usually made on the basis of comparative tests.[13] Important contributions in quantifying machine performance have been made by Rose[15] and by Myers and Lewis.[14] The rate approach offers the possibility of integrating performance with capacity, and of providing a single number index for comparing machines.

For a separation in any machine the rate constant as already indicated will depend on the ore and its pretreatment as well as on the machine. The machine contribution cannot be isolated for the single system. However, if the identical pulp is treated in another machine, the ratio of rate constants will be a relative measure of the contribution of each machine to the rate. Thus copper ore *A* gave a constant of 2.5 in machine 1, and of 2.0 in machine 2. For machine 1, treating ore *A* in a laboratory batch test, the rate constant was 2.6; for mill cells of the same design, treating similar but not identical pulp, the constant was 2.9.

If, for simplification, it is assumed that the maximum possible recovery with a given pulp is close to unity, eq 3 becomes

$$Kt = R/(1-R)$$

Since tonnage capacity is inversely proportional to the flotation time requirement, then at a fixed recovery level the capacity will be directly proportional to the rate constant. Hence the ratio of rate constants for the same pulp in two different machines will be equal to the ratio of capacities obtainable with the two machines for the same recovery. On the other hand, the ratio of rate constants for constant tonnage through the two machines on the same pulp will be related to differences in recovery.

Machines also may be rated according to selectivity rather than according to the rate for one mineral. In laboratory flotation of ore *A*, the copper/iron selectivity was 3.6 for machine 1, and 5.0 for machine 2, indicating that the first machine makes a higher extraction at any time but at the expense of lower selectivity.

Capacity, Power, and Rate Constant

It has been indicated that the second order rate equation where valid leads to a direct proportionality between rate constant and cell capacity at a fixed recovery level. Rose has shown[15] that cell capacities within a group of 14 operations varied in an approximate linear manner with power intensity (horsepower per cubic foot of cell volume). For a first approximation, as Rose suggested, power intensity is more important in determining rate than ore, ore treatment, and machine design. However, the deviations from the linear relationship, and the fact that in any one machine differences in rate constants for different ores are obtained, suggest that the secondary effects cannot be neglected in a complete analysis. The role of power intensity in controlling the rate constant can be related to its effect of power on aeration and on sand suspension.

Aeration: For mechanical cells an increase in power intensity caused by increased impeller speed results in increased aeration, although not at a constant rate (unpublished data). A similar condition should exist for pneumatic cells. Total air input, however, is not a sufficient measure of the effectiveness of air in flotation, since bubble size must be considered. Cooper[16] has shown that the efficiency of air in oxidizing sodium sulphite solutions, which depends on both total air volume and bubble size, is directly proportional to agitator power intensity at constant air input. Similarly, Foust[17] found that bubble dispersion, with constant air input to an agitated vessel, varied with the square root of power intensity.

For flotation cells, Myers and Lewis[14] report aeration intensity and power intensity data which indicate a fourfold variation in aeration per unit of power between the extreme values. Thus, while aeration quantity and quality may depend principally on power intensity, machines will differ in the efficiency of power utilization for aeration.

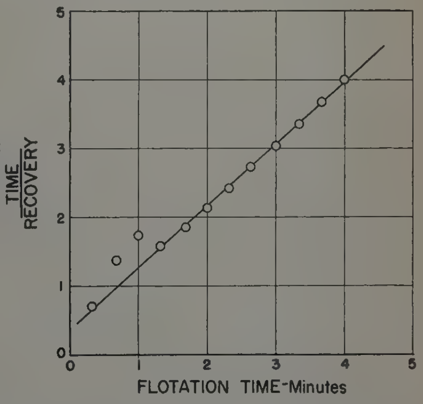

Fig. 7—Graphical test of second order rate equation. Continuous flotation, low grade copper ore.

Sand Suspension: Sand suspension by mechanical agitation has been studied to develop a criterion for agitation intensity.[17] The principal findings were that in the size range 32 to 100 mesh suspension depends quantitatively on sand size and impeller speed. For any size in this range increased impeller speed progressively increases suspension until it is complete. The speed necessary for complete suspension increases directly with sand size. It was also found (and confirmed in this investigation on an operating flotation cell) that suspension is not necessarily uniform, even though complete. Although no comparative data on sand suspension for flotation cells are available, it is to be expected that cells will differ in this respect just as with respect to aeration.

Flotation Efficiency

The efficiency of a flotation cell is related to the return in aeration and sand suspension per unit of power expended. No general relationship is known between the flotation result on the one hand and a measure of either aeration or sand suspension on the other. Moreover, increased aeration and sand suspension are useful only to the extent that performance continues to improve. The rate constant, therefore, which is a direct measure of performance, is a logical index of efficiency when related to the power expended. In other words, one machine is more efficient than another when it develops a higher rate constant for the same energy expenditure on the same pulp.§§

Summary

It is shown that flotation results can be analyzed in terms of rates, and that a rate constant in a second order rate equation is a useful measure of performance. The ratios of such constants can yield relative measures of the contributions of both pulp and machine to the rate. The ratio of the rate constant to the energy expenditure is proposed as a measure of flotation efficiency.

This approach to the analysis of the flotation process is based on the hypothesis that a mineral has a potential floatability determined by its previous treatment and that the machine realizes this floatability through the medium of pulp suspension and aeration. The floatability of the mineral cannot be defined precisely except in terms of the actual flotation result; similarly, the machine contribution, although related to power intensity within limits, is best defined in terms of the operation. The further analysis of rates and determination of rate constants should aid in clarifying these relationships and in advancing the efficiency of the process.

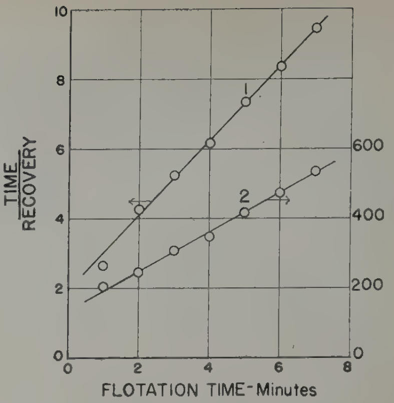

Fig. 9—Graphical test of second order rate equation. Continuous flotation, molybdenite from low-grade concentrate.

1—Molybdenite
2—Copper

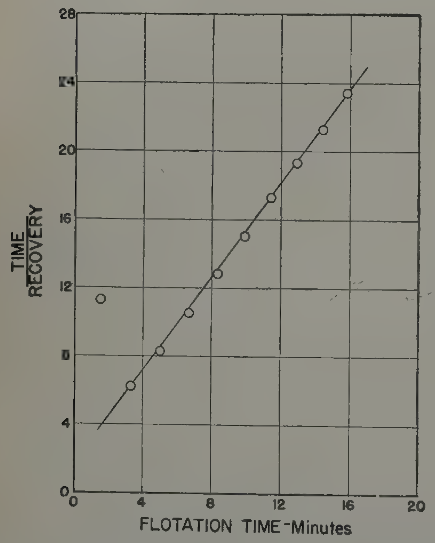

Fig. 8—Graphical test of second order rate equation. Continuous flotation, sphalerite from massive sulphide ore.

Acknowledgments

Appreciation is expressed to W. H. Osborn, Director of Development and Research, Phelps Dodge Corp. for encouraging this work and permitting its publication, to J. F. Myers and F. M. Lewis of the Tennessee Copper Co. for furnishing the sphalerite rougher data, and to numerous associates for encouragement and suggestions.

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§§ This index will have the dimensions: Recovery-tons /hour /horsepower hour if the customary horsepower hours/ton is used to relate tonnage with energy.

Magnetic and Chemical Analyses Of Ores and Mill Products Containing Magnetite and Ilmenite

by Erkki Laurila, O. Jantti, and R. T. Hukki

INVESTIGATION of the methods of analyses for magnetite and ilmenite in the Otanmäki iron-titanium ore and respective mill products has resulted in certain improvements in the methods conventionally employed. The outcome of the investigations is summarized as follows:

1—Development of a diamond-drill core analyser for the determination of magnetic susceptibility of drill cores containing magnetic minerals.

2—Development of a magnetite-ilmenite analyser for pulverized products.

3—Development of a potentiometric method of chemical analysis for the determination of iron, vanadium, and titanium in succeeding steps from one test sample.

4—Proposal for a continuous and automatic magnetite assay method.

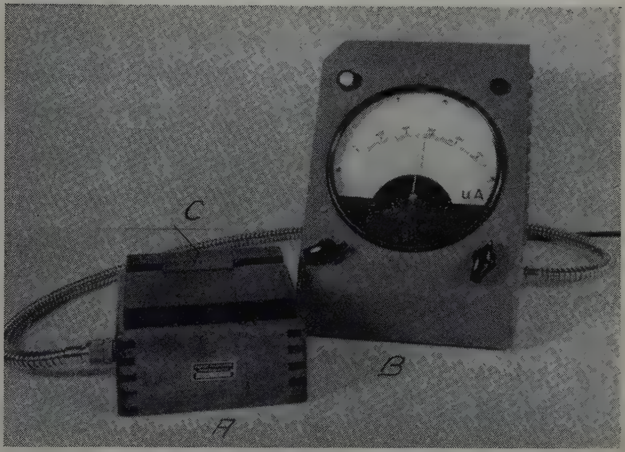

Fig. 1—Diamond drill core analyser.

Diamond-Drill Core Analyser

The conventional method of assaying diamond-drill cores consists of splitting the cores, pulverizing them and running chemical analyses on representative samples. The physical properties of magnetic minerals, however, offer an entirely different approach to obtain similar final result by the measurement of respective magnetic susceptibilities in a properly designed instrument.

This instrument is shown in Fig. 1. Essentially it consists of a magnetic unit, *A*, and of a meter unit, *B*. On top of *A* is a slot where a piece of diamond-drill core *C* is visible. The magnitude of disturbance in the original magnetic field caused by the magnetic minerals in the drill core can be observed from the microammeter. This reading is a function of the combined susceptibilities of all magnetic grains in the magnetic field.

The magnetic unit is built of E-sheets of a shell transformer, as shown in Fig. 2. The primary coil and two identical secondary coils are wound around the central leg and end legs, respectively. The drill core under examination is placed into one of the air gaps as shown in Fig. 2. The number of ampere-turns of primary coil excited by 50 or 60-cycle ac give a maximum induction of about 600 gauss in the air gaps. The secondary coils are connected in series in the opposite direction. Thus their voltages compensate each other if no drill core exists in the air gap or if the susceptibility of the drill core is equal to zero. As soon as the susceptibility becomes greater than zero, the balance of the magnetic circuit is disturbed, resulting in increased voltage in the terminals of coil 1 and in a respective decrease of voltage in the terminals of coil 2. The voltage difference thus produced is a function of the effective susceptibility of the drill core. If the magnetic grains in the drill core are unevenly distributed, the voltage difference varies as the core is rotated around its own axis.

The voltage difference is measured with a Brentano tube voltmeter. If extreme sensitiveness is desired, the established symmetry of the magnetic circuit does not allow reduction of the O-voltage difference far enough. Therefore a compensatory voltage of a correct phase taken from the terminals

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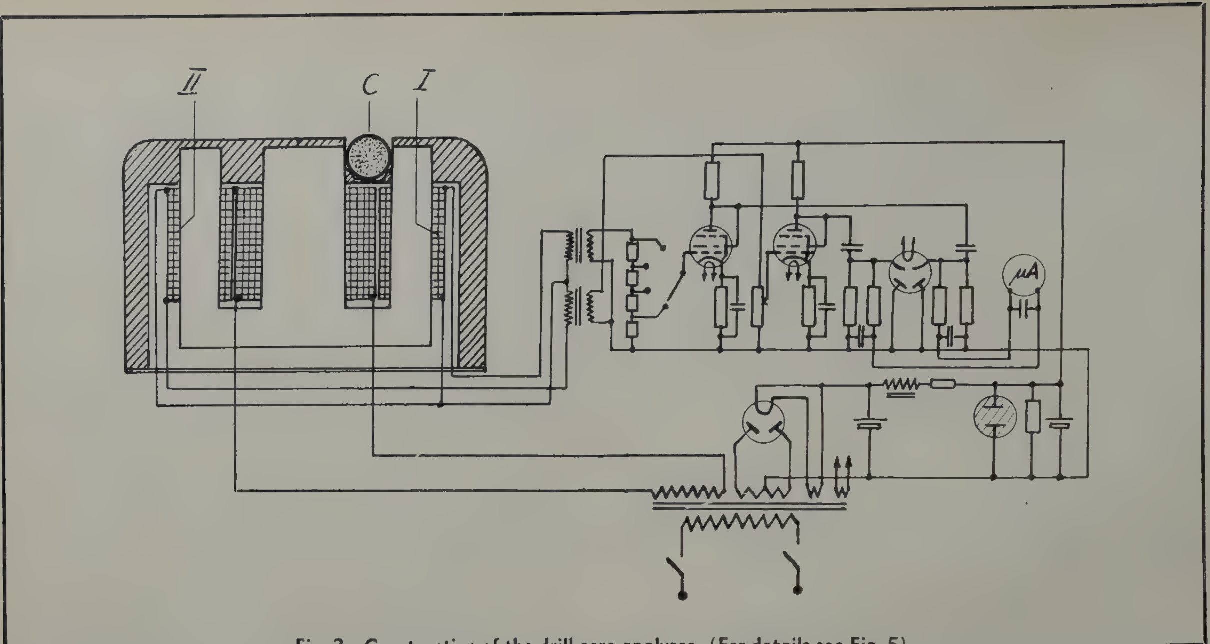

Fig. 2—Construction of the drill core analyser. (For details see Fig. 5)

of one of the secondary coils is connected to the grid of one tube of the tube voltmeter. With this arrangement it has been possible to include four scales of sensitiveness in sequence of 1:4:16:64. The least sensitive scale is used for cores containing magnetite. A core 22 mm in diam of relatively pure magnetite causes a reading of about 80 pct of the available scale. The lowest value of susceptibility that can be observed with the most sensitive scale is of the order of 0.001.

The electronic system of the drill core analyser is shown in Fig. 2.

To test the instrument, a box containing one drill core representing the Otanmäki ore and wall rocks was received. Nine samples were selected from it to cover as wide a range of analyser readings as possible. Two cores representing O-reading were selected because of differences in mineral composition. Each sample core selected was tested in the analyser with the least sensitive scale. Readings were taken in 5-mm steps between two chosen end marks. For each reading the core was rotated 360°, the maximum and minimum readings were recorded and averaged. Their average was taken as the final reading for the whole sample. The cores were cut at the end marks, pulverized, and assayed by the chemical method described later. Table I indicates the results obtained. Analyser readings vs. total iron contents are shown in Fig. 3.

Table I. Magnetic and Chemical Analyses of Samples of a Diamond-Drill Core

Drill Core	Analyser Readings			Total Fe, Pct			TiO_2, Pct		
	Max	Min	Avg	a	b	Avg	a	b	Avg
1	0	0	0	13.25	13.25	13.25	5.89	6.01	5.95
2	0	0	0	12.87	13.11	12.99	6.14	5.70	5.92
3	10	30	20	15.32	15.41	15.36	1.57	1.50	1.53
4	40	80	57	23.19	23.09	23.14	7.46	7.36	7.41
5	65	125	94	28.40	28.47	28.43	9.48	9.34	9.41
6	125	190	162	34.77	34.40	34.58	10.51	10.64	10.57
7	110	270	218	43.28	43.10	43.17	17.38	17.28	17.33
8	205	260	239	45.55	46.08	45.81	15.41	15.42	15.41
9	225	270	255	48.55	48.91	48.73	15.64	16.12	15.83

Table II. Susceptibilities of Otanmaki Magnetic and Ilmenite

Max Field Strength, Amp-Turns per M	Susceptibility of Magnetite	Susceptibility of Ilmenite
500	2.80	
5000	3.35	0.025
25000	2.60	0.030
50000	2.05	0.022
100000		0.016

As can be seen from Fig. 3, the points fall nearly on a straight line. Whether the relationship follows a straight line or a curve depends on the type of ore. Should magnetite be the only iron-bearing mineral, a straight line relationship is to be expected. In the case of Otanmäki ore the analyser reading on the abscissa is affected slightly by the presence of ilmenite, and the value of total iron on the ordinate includes in addition to magnetite such iron-bearing minerals as ilmenite, pyrite, hornblende, etc. Their distribution in the ore does not depend on that of magnetite. Before the true average shape of the graph in this case can be presented with certainty, a substantial number of experimental values is required.

The analyser offers a rapid and inexpensive method for preliminary analysis of drill cores carrying magnetic minerals. It obviously is best suited for ores where magnetite is the only iron-bearing mineral. No great accuracy can be claimed mainly because of unavoidable variations in core diameter. Since the diamond-drill core itself, even with the best possible recovery, represents only an approximation of the respective rock; and the splitted halves are an approximation of the cores recovered; the drill-core analyser method, which takes into account the entire core available (if readings are taken in close enough steps) probably represents the same order of accuracy as can be claimed for the conventional method. In field use, the analyser can reduce materially the number of chemical assays required, but it is not expected to replace them altogether. Results of magnetic and chemical analyses of samples of a diamond-drill core of Otanmäki are given in Table I.

Magnetite-Ilmenite Analyser

In assaying pulverized products containing magnetite, the use of a magnetic analyser such as the Davis magnetic tube tester is a well-known and widely accepted practice. In the Davis instrument the magnetic particles of a weighed sample are held inside a water-filled glass tube by a strong dc electromagnet, or more recently by a permanent magnet, while the nonmagnetic particles are thoroughly removed by the combined actions of gravity, reciprocation of the tube, and wash water. Either the magnetic fraction retained or the nonmagnetic fraction removed is dried and weighed to obtain the amount of magnetite in the original sample.

In the case of Otanmäki ore it was desirable to develop an instrument in which the physical evaluation of magnetite content of pulverized samples could be followed by a similar evaluation of ilmenite content of at least such products as ilmenite concentrate and ilmenite tailing.

The physical evaluation of magnetite and ilmenite is based on their magnetic properties. The susceptibilities of carefully prepared —100 mesh samples of high grade Otanmäki magnetite and ilmenite measured in ac magnetic fields are shown in Table II.

Two methods for the evaluation of magnetite and ilmenite of the same sample are possible, theoretically at least. One of them could be based on two determinations of susceptibilities of a sample in two magnetic fields of widely different strengths. The other could be based on magnetic separation of magnetite and ilmenite into separate fractions. The method described here is based on the second principle. The evaluation of the amount of magnetite and ilmenite in the respective fractions is done by electrical instruments.

Model No. 2 of the analyser developed is shown in Fig. 4 and its construction in Fig. 5. Essentially the instrument consists of a magnetic unit, *A*, and a meter unit, *B*.

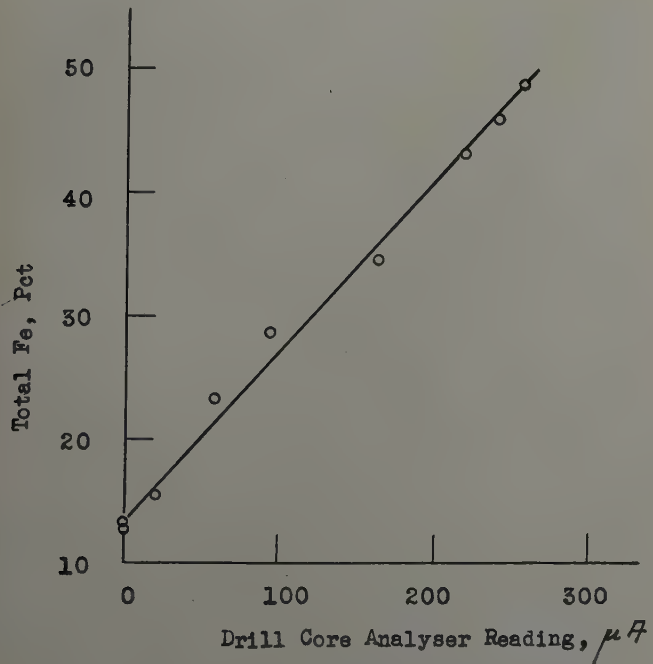

Fig. 3—Plot of drill core analyser readings vs. total iron content.

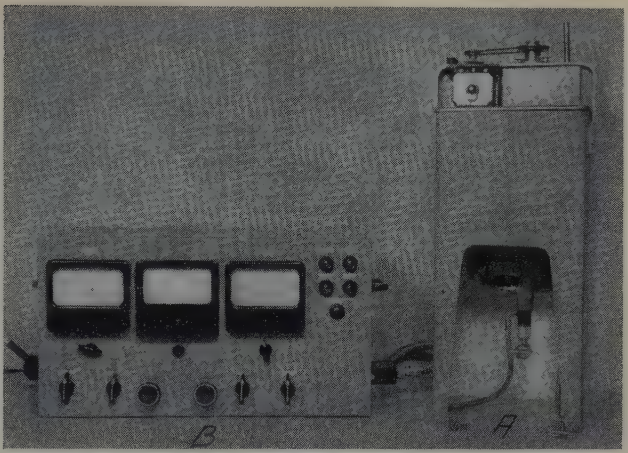

Fig. 4—Magnetite-ilmenite analyser.

A pulped sample is introduced through the funnel of a slowly revolving distributor into a vertical glass tube filled with water. The magnetite fraction of descending particles is held against tube walls by an inhomogeneous magnetic field produced by an ac excited magnet resembling in construction a "magnetic lens." The strength of the field holds all particles of magnetite but no ilmenite. The tube is surrounded also by a secondary coil in which the retained magnetite induces a voltage the value of which is a function of the amount of magnetite present. This voltage is indicated in the meter unit by a tube voltmeter.

The ilmenite fraction is held by a strong ac magnetic field of the type used in the drill-core analyser described above. The ilmenite retained causes a change in the flux of the magnetic field resulting in a change in the voltage difference of secondary coils.

Because the voltage differences in the magnetite and especially in the ilmenite coils are small and the impedance of the coils relatively large, the voltages are measured by tube voltmeters. The meter in the magnetite circuit is a tube voltmeter of Brentano type. The voltage to be measured is connected to the grid of one tube and a compensatory voltage from a separate transformer to the grid of the other tube of the voltmeter. Thus the indicated reading is equal to zero if no magnetite exists in the analyser tube. In the ilmenite circuit this arrangement is not satisfactory, but a two-stage tube voltmeter is necessary. The first stage is the same as described above in connection with the drill-core analyser. Its output is rectified by a diode circuit and is connected to the second stage operated as a dc amplifier. To obtain sufficient stability, all amplifier circuits have a strong negative feedback. The complete wiring diagram of the magnetite-ilmenite analyser is shown in Fig. 5.

To test the instrument (Model No. 2), two samples of —100 mesh high grade magnetite and ilmenite were used. For magnetite the upper curve shown in Fig. 6 was obtained. The nonlinear scale is intentional because greater accuracy was desired for small amounts of magnetite. The lower curve in Fig. 6 represents similar readings for ilmenite samples containing no magnetite. Because specially prepared clean samples were used, the above calibration curves are of theoretical value only. They clearly show, however, the basic characteristics of the instrument.

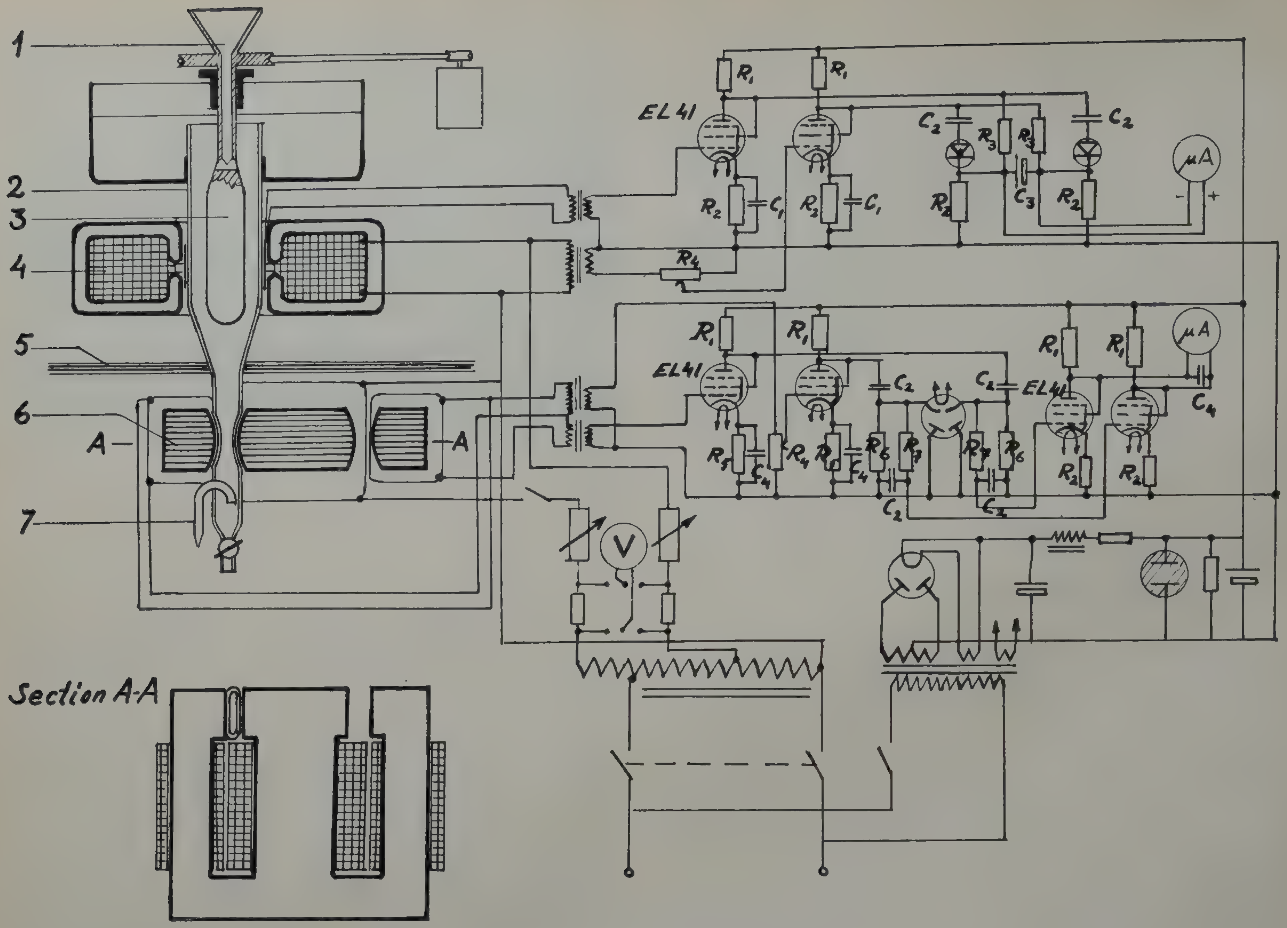

Fig. 5—Construction of magnetite-ilmenite analyser.

1. **Funnel**
2. **Glass Tube**
3. **Distributor**
4. **Magnetic circuit for magnetite**
5. **Magnetic shield**
6. **Magnetic circuit for ilmenite**
7. **Discharge tube**

R_1 — 40 k Ω
R_2 — 5 k Ω
R_3 — 70 k Ω
R_4 — 1 k Ω
R_5 — 2.5 k Ω
R_6 — 200 k Ω
R_7 — 65 k Ω

$C_1 = 50\mu F$
$C_2 = 0.5\mu F$
$C_3 = 100\mu F$
$C_4 = 50\mu F$

Actual mill products of Otanmäki ore all contain a mixture of varying amounts of magnetite, ilmenite, and a score of nonmagnetic gangue minerals. To meet the practical conditions of everyday drift samples, a new set of calibration curves must be prepared including all classes of samples that are to be analysed by this method.

Our experience indicates that no particular difficulties were encountered in the magnetite analyses of actual mill products. As a matter of fact, Model No. 1 of this instrument has operated satisfactorily for several months in the Otanmäki pilot plant. The presence of ilmenite or gangue or locked particles of any kind within the magnetite retained is not a disturbing factor in magnetite determinations because the susceptibilities of other minerals are small compared with that of magnetite. If they have an effect, it is small and consistent within each class of samples.

A correct determination of ilmenite simultaneously with magnetite requires that all ilmenite particles report in the ilmenite circuit. This condition hardly can be fulfilled in analysing magnetite concentrates containing small amounts of ilmenite. The interlocking of even a small fraction of the ilmenite within the large bulk of magnetite is enough to cause serious errors in the amount of ilmenite reporting in the lower field. With ilmenite concentrate and ilmenite tailing samples containing little or no magnetite, the conditions stated above can be met either without any precautions or with a pretreatment of the sample to be analysed in a small additional ac magnetic separator unit of such construction as to allow the magnetite-free fraction to fall into the analyser tube ahead and the substantially clean magnetite fraction immediately behind.

The magnetite-ilmenite analyser developed has the following advantages when compared with the conventional magnetic tube tester method: 1—Two magnetic minerals of different susceptibilities can be separated for physical evaluation. 2—The ac magnetic field used cause a strong cleaning effect through the intense vibration of the magnetic particles in the field. 3—The amount of magnetic mineral reporting in each circuit is evaluated by electrical instruments. 4—The instruments do not record the presence of gangue minerals whether in free or locked particles. 5—The final results are obtainable within a minute or two.

Potentiometric Analysis

The simultaneous presence of iron, vanadium, and titanium in mill products of Otanmäki ore has introduced certain difficulties especially in the accurate chemical determinations of vanadium and titanium.

Two methods[1, 2] have been recently described by which iron and titanium can be volumetrically determined from one solution. No description including all three was discovered. The method presented here for the determination of iron, vanadium, and titanium is primarily based on the paper of Lingane[2].

Principle: A sample containing V^{+++++}, Fe^{+++} and Ti^{++++} ions in sulphuric acid medium is titrated with a titrant of chromous sulphate. The change of potential indicated by a potentiometer plotted against titrant added gives a curve shown in Fig. 7. In it the first major fall represents reduction of V^{+++++} to V^{++++}, the second one simultaneous reduction of Fe^{+++} to Fe^{++} and V^{++++} to V^{+++}, and the third one reduction of Ti^{++++} to Ti^{+++}. The amount of vanadium in the original sample is obtained by evaluation of titrant used for the first fall of potential up to the first inflexion point in the curve; this reading is multiplied by factors related to the normality of titrant and to the chemical formula with which the final result is to be expressed. The amount of iron in the sample is obtained similarly by evaluation of the amount of titrant used for the second fall of potential between the first and second inflexion points subtracted by a correction factor equivalent to the amount of titrant used for the reduction of V^{+++++} to V^{++++}. This correction takes care of the reduction of V^{++++} to V^{+++}. The difference is multiplied by respective factors for iron for final result. The amount of titanium is obtained by evaluation of the amount of titrant used between the second and third inflexion points and by multiplying this value with respective factors. The paper of Lingane[2] may be consulted for detailed physicochemical discussion of the method for iron and titanium as well as for apparatus necessary to automatice the procedure.

Apparatus: The apparatus used is shown in Fig. 8. It consists of two identical systems to allow simultaneous titration of one sample and removal of oxygen by CO_2 gas of another sample. Titration takes place under CO_2 atmosphere. The change of potential is observed from a potentiometer not included in Fig. 8.

Procedure: A 400-mg sample is mixed in a nickel crucible with 5 g of Na_2O_2 and fused carefully for 10 min. The hot crucible is dipped halfway into water and placed upside down on top of glass sprinkler, *11* (Fig. 8), in a 400-ml beaker. Through the sprinkler is poured 60 ml of boiling water followed by 50 ml of hot sulphuric acid (1 + 1). The crucible is removed. A few drops of 3 pct H_2O_2 are added to dissolve precipitated MnO_2. Solution is boiled for 5 min and cooled down to room temperature. The beaker is placed for titration position and covered tightly. Oxygen-free CO_2 is bubbled through the solution for 20 min and continued during titration to remove oxygen. Vanadium and iron are titrated by using platinum indicator electrode against calomel reference electrode. When equivalence point potential is reached, the platinum electrode is lifted out of solution, the mercury indicator electrode is inserted, and operation is continued beyond the third inflexion point.

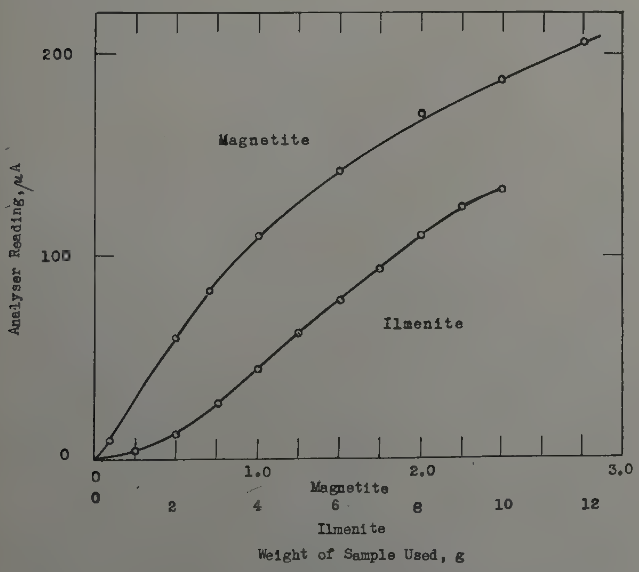

Fig. 6—Plot of calibration curves of magnetite-ilmenite analyser.

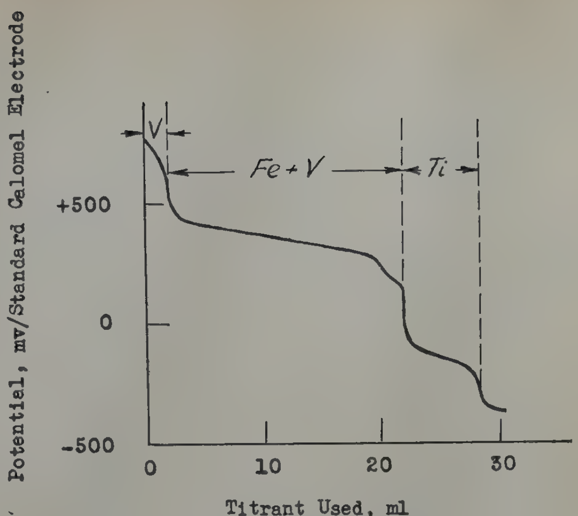

Fig. 7—Potentiometric titration of +3 iron, +5 vanadium and +4 titanium with chromous ion.

Reagents: Amalgamated zinc is prepared by treating 1 kg of analytically pure zinc for 10 min in a solution consisting of 28 g of $HgCl_2$, 100 ml of HCl (1.19) and 500 ml of water. After the treatment the zinc is washed free of chlorides. The storage flask *2*, (Fig. 8), is fully filled with the amalgamated zinc thus obtained.

The titrant is prepared in a 1½ to 2-liter conical beaker by dissolving 38.84 g of $K_2Cr_2O_7$ in 200 ml of water, after which 65 ml of H_2SO_4 (1.84) and finally 75 ml of 30 pct H_2O_2 are carefully added. Solution is kept boiling for 30 min, cooled down and diluted in a volumetric flask to 2-liter volume. The storage flask, *2*, is filled with this solution 8 to 12 hr before use to allow time for complete reduction to chromous ion. With the purest zinc available the titrant could be preserved for a maximum of 1 week.

Wash solution of CO_2 is prepared by the same method as the titrant. It is diluted, however, to 500 ml only. Zinc in 200-ml wash bottles, *1* (Fig. 8), is covered with this solution about 12 hr before the apparatus is used. If CO_2 gas contains less than 0.2 pct by volume of oxygen this method of deoxidation is satisfactory.

The chromous sulphate titrant is checked against 0.1 molar $CuSO_4$ reference solution, of which 25 ml are mixed with 25 ml of water and 50 ml of HCl (1.19). The exact copper content of the reference solution is determined by electrolysis.

Electrodes and their care: A satisfactory platinum electrode consists of an 80-mm piece of 1-mm plati-

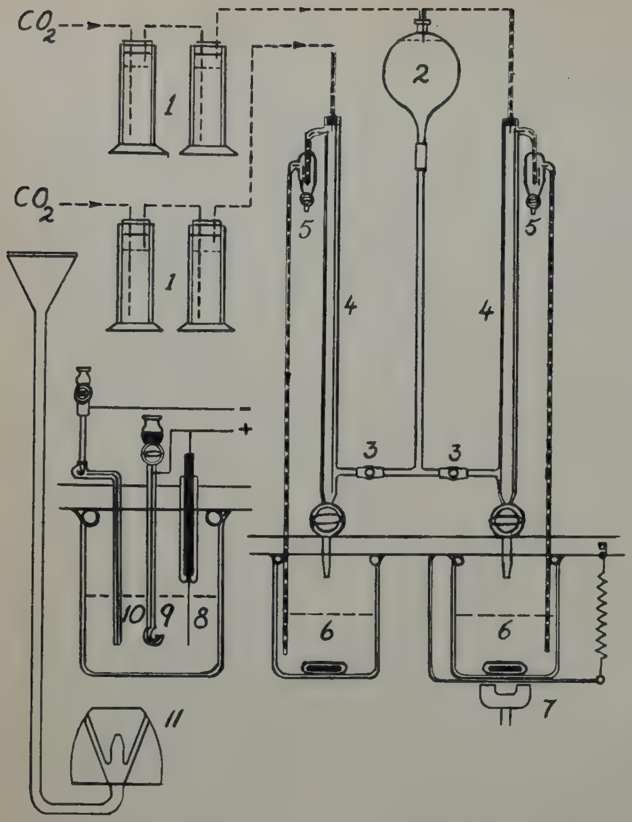

Fig. 8—Apparatus for potentiometric analysis of iron, vanadium, and titanium.

1. **Wash bottles for removal of oxygen of CO_2 gas**
2. **Titrant storage flask**
3. **Glass bead valve**
4. **Burette**
5. **Removal of excess of titrant**
6. **Titration vessel**
7. **Magnetic stirrer**
8. **Platinum indicator electrode**
9. **Mercury indicator electrode**
10. **Calomel reference electrode**
11. **Sprinkler for removal of fused sample**

num wire soldered to a thin brass rod. The rod is inserted in a glass tube permanently connected to the cover of titration vessel. The electrode thus can be removed easily for washing in HCl and ignition to secure its absolute cleanliness.

The construction of the mercury electrode is shown in Fig. 8. Analytically pure and gas-free mercury must be used. The platinum contact with mercury should be occasionally cleaned.

Discussion of Method

The potentiometric analysis of iron, vanadium, and titanium has given very satisfactory service in the chemical assaying of products of Otanmäki iron-titanium ore. As a matter of fact, the three constituents can be determined faster by this method than by any other method known to the authors. One determination from a pulverized sample requires normally 45 to 60 min. With the apparatus described above the (absolute) errors seem to be within the following limits (see also Table I): Fe, ± 0.3 pct; V_2O_3, ± 0.1 pct; TiO_2, ± 0.2 pct.

Because the amount of vanadium (V_2O_3) in all products assayed has been less than 1 pct, the (absolute) error of ± 0.1 pct is large. Vanadium can be determined with greater accuracy from a substantially larger initial sample. If a 5 g weighing is used (40 g Na_2O_2 for fusion and 100 ml 1 + 1 H_2SO_4 for dissolution), then the (absolute) error of V_2O_3 may be reduced to ± 0.02 pct.

Continuous Magnetite Assay

A fully automatic and continuous plant assay control system is the dream of many a millman. Numerous attempts have been made to solve the problem, but as far as the authors know none of them has been successful.

The magnetic instruments described in this paper seem to offer a marked step forward toward continuity, instantaneous evaluation, and exact recording of product assays. The following apparatus seems to be essential for the determination of magnetite:

1—Equipment to form continuous and representative sample bar of the pulverized product to be analysed. This bar corresponds to a diamond-drill core in the preceding discussion.

2—Magnetic unit through which the sample bar moves. In principle this unit is the same as the one used for diamond-drill core analyser.

3—Measuring and recording unit for the evaluation of the magnetic effect caused by the sample bar. The measuring unit would correspond to the respective unit of the drill core analyser. The recording instrument could be of the general type used in continuous recording, e.g. Micromax of Leeds & Northrup Co., Brown Electronik Potentiometer of Minneapolis-Honeywell Regulator Co.

No attempt has been made to put the necessary apparatus together. Cutting of a representative sample and formation of a uniform sample bar may prove to be the basic problem in the procedure suggested.

The products that could be assayed by the method described above include all powdered products of magnetite concentrators where magnetite is the only magnetic mineral. In the case of Otanmäki ore, where both magnetite and ilmenite occur in varying amounts in all products, the continuous assaying of products rich in magnetite should be possible. The adaptability of the method for magnetite tailing assays may or may not be feasible.

A combination of a continuous magnetic assay system with a continuous magnetic weightometer integrator system indicating the total effect of all magnetic particles in the concentrate produced would result not only in automatically recorded quality but also in automatically recorded quantity of the magnetite concentrate.

Acknowledgments

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Characteristics of Mechanized Mining Sections

by A. W. Asman and A. W. Bitner

An analysis is made of three different types of section production units that represent the manner in which most of the nation's bituminous coal is produced. The general delays and production characteristics of these sections are presented along with a method for evaluating section performance based on actual and theoretical production characteristics.

THE forerunner of the modern mechanical loading machine first entered the mines in 1890—a Stanley header with a conveyor. Since then progress has been substantial, both in the design and application of mechanized production units. Mechanization is almost a necessity for survival in the economy of today.

In this paper an analysis is made of three types of sections, and a method for evaluating section performance based on present production characteristics and theoretical characteristics is presented.

In mechanical loading mining sections the loading machine is the unit directly responsible for production. If the face preparation units provide a plentiful supply of properly prepared coal and the transportation system is coordinated and adequate, the loader will be idle a minimum of time. If either of the above do not perform adequately or other delays occur, the effect on production will be reflected in the performance of the loading machine.

Therefore, attention is focused on the loading machine and factors that affect the overall performance of the production unit. Three complete mechanized production units are presented for analysis: A track section, a trackless section using battery-powered shuttle cars, and a trackless section using cable-reel shuttle cars. These particular sections were chosen for the following reasons: 1—The daily tonnage and mining conditions are approximately the same. 2—Production in each is 10 to 20 pct above the daily average. 3—The production characteristics and delays are average and typical. 4—They represent the manner in which the most of the nation's bituminous coal is produced.

Section A

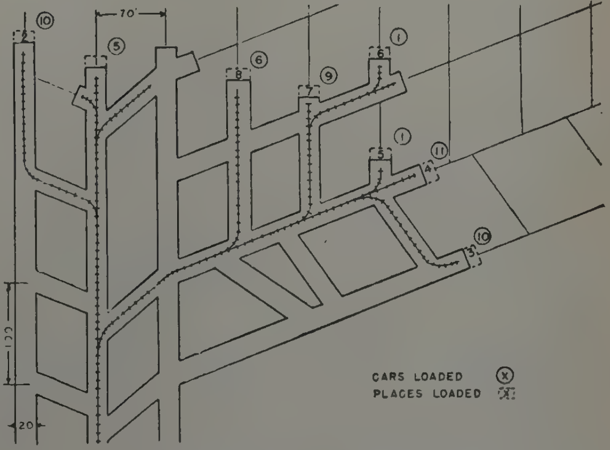
Fig. 1 is a layout for the track mining section showing the development to date, the projected plan and the cuts loaded on the shift studied. The seam mined averages 96 in. in thickness with a fireclay bottom and a roof of 10 in. of drawslate topped by sandstone. The mineability is considered good, and roof support is not a difficult problem. Two locomotives are used in the section to minimize the losses occurred in trip changing. The places are driven 20 ft wide, and the crosscuts are angled to permit fast tramming of the equipment.

Table I lists the equipment and labor that make up the section.

ANALYSIS

Production Characteristics: The average loading rate for eight loading machines studied of this type is 1.81 tons per min. This one at 2.78 is considerably above average.

Fig. 1—Section A track equipment.



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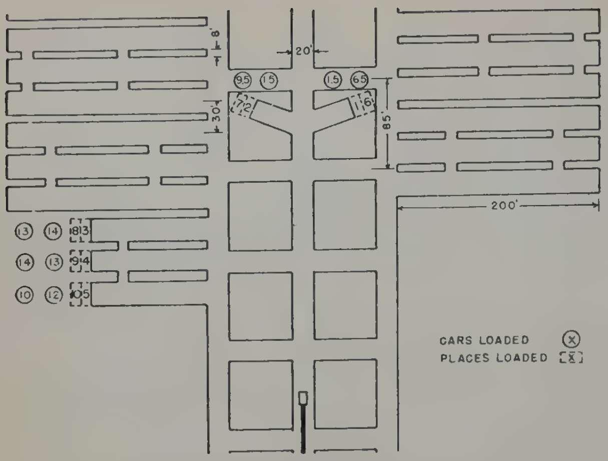

Fig. 2—Section B trackless equipment, battery shuttle cars.

Delay Characteristics: Mechanical delays accounted for 20.58 pct of the shift. Careful handling and preventative maintenance can reduce this item. Derails totaled 5.75 pct, a product of rough handling, poor track work, or operating technique. Inefficient deadwork and face preparation accounted for 6.15 pct of shift. Miscellaneous delays were low at 1.17 pct. Two places loaded yielded only one car each. Proper cleanup of these cuts on the initial loading cycle would have eliminated the tramming time involved.

Section B

Fig. 2 is a layout of a trackless mining section using battery-powered shuttle cars.

The seam mined averaged 72 in. in thickness.

The roof varies from 0 to 40 ft of friable shale and extensive timbering is necessary; the bottom is of hard fireclay. Mineability is considered fair with some difficulty in shooting caused by a middle band of rock.

Rooms are mined in sets of three on retreat approximately 24 ft wide on 30-ft centers. Entry pillars are drawn at the same time. No attempt is made to recover the room pillars.

Table II lists the equipment and personnel in the section.

ANALYSIS

Production Characteristics—The average loading rate for 23 loading machines studied of this type is 2.27 tons per min. At 1.96 tons per min this loader is below average, probably because of top rock and binder in the coal. Place change is about average for a crawler type loader. Travel distance of 624 ft per round trip is below the average of 850 ft for battery cars. Travel speed of 275 ft per min is less than the 326 ft per min average for battery cars, and discharge rate is average.

Delay Characteristics—(Loader) Inefficient deadwork and face preparation caused the largest delay, 10.80 pct; lack of coordination of main haulage with no empty mine cars, 4.38 pct; mechanical delays, 3.63 pct; and roof fall on loader, 1.96 pct. Miscellaneous delays were low at 1.92 pct.

Table I. Section A

Equipment: Loading machine, rated 3 tons per min; Universal cutting machine, bar 9 ft bottom cutting; drilling machine, 6 holes per cut; two 8-ton locomotives, 3 car trips; steel mine cars, rated 6.5 tons capacity.

Personnel: Section foreman, Loading machine operator, Loading machine helper, Cutting machine operator, Cutting machine helper, Driller, Shot firer, Timberman, two Trackmen, two Motormen, two Brakemen—**Total, 14.**

Operating Data	
Mining height, in.	96
Production, tons	326
Number of cars	53
Average car capacity, tons	6.15
Number of men	14
Production per man-shift, tons	23.2
Time in section, min	425

OPERATING CHARACTERISTICS	
Production Time, 66.35 Pct	
Average loading time per car	2.22
Average change car time per car	2.68
Total per car	4.90
Average place change time per place	2.81
Average trip change time	6.13
Average loading rate, tons per min	2.78
	282.00

TIME ELEMENTS OF SHIFT STUDIED	Min
Production Time, 66.35 Pct	
Loading time, 27.64 pct	117.45
Change car time, 33.42 pct	142.05
Place change time, 5.29 pct	22.50
	282.00
Delay Time, 33.65 Pct	
Tram chain failure (2), 12.92 pct	54.90
Loader derailment (2), 5.14 pct	21.85
Controller adjustment, 3.16 pct	13.45
Slate and lump handling, 2.70 pct	11.45
Lubrication, 2.44 pct	10.35
Locomotive cable failure, 2.06 pct	8.75
Track extension, 1.98 pct	8.40
Move junction box, 1.47 pct	6.25
Car derailment (3), 0.61 pct	2.60
Miscellaneous, 1.17 pct	5.00
	143.00
Total	425.00

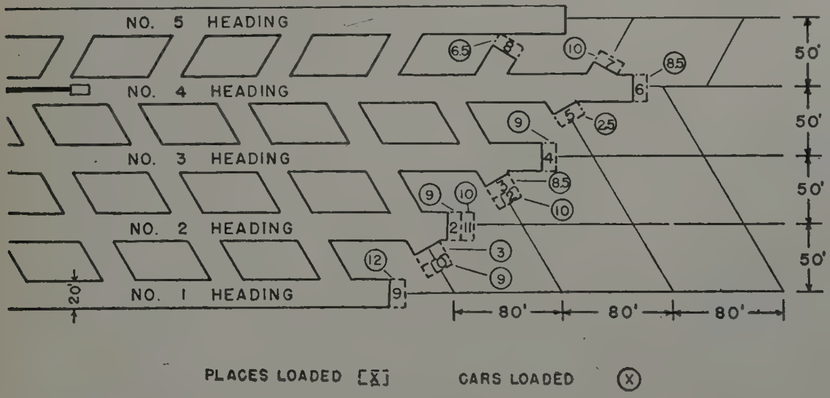

Fig. 3—Section C trackless equipment, cable shuttle cars.

Table II. Section B

Equipment: Crawler-mounted loading machine rated 5 tons per min, two battery-type shuttle cars rated 3.75 tons, Crawler-mounted cutting machine (9-ft bar, center cutting), Hand-held drill (seven holes per cut), 30-in. belt conveyor.

Personnel: Section foreman, Loading machine operator, Cutting machine operator, Cutting machine helper and driller, Shot firer, two Shuttle car drivers, two Timbermen, Supply and timberman, Boom operator—**Total, 11.**

Operating Data

Mining height, in.	72
Production, tons	332
Number of shuttle car loads	95
Average shuttle car capacity, tons	3.50
Number of men	11
Production per man shift, tons	30.2
Time in section, min	434

OPERATING CHARACTERISTICS

Loading Machine	Min
Production Time, 77.31 Pct	
Average loading time per car	1.79
Average change car time per car	1.31
Total per car	3.10
Average place change time per place	4.10
Average loading rate, tons	1.96
	335.50

Shuttle Cars	1	2
Trips	49	46
Travel speeds, ft per min	274	275
Average round trip, ft	618	624
Discharge time per trip, min	0.75	0.82
Discharge rate, tons per min	4.76	4.35

TIME ELEMENTS OF SHIFT STUDIED

Loading Machine	Min
Production Time, 77.31 Pct	
Loading time, 39.09 pct	169.65
Change car time, 28.78 pct	124.90
Place change time, 9.44 pct	40.95
	335.50
Delay Time, 22.69 Pct	
Move junction box (2), 6.15 pct	26.70
No empty mine cars, 4.38 pct	19.00
Loader cable, 3.36 pct	14.60
Slate and lump handling, 2.90 pct	12.60
Fall on loader, 1.96 pct	8.50
Timbering, 1.75 pct	7.60
Power failure, 0.27 pct	1.15
Miscellaneous—shuttle car, 0.99 pct	4.30
Miscellaneous, 0.93 pct	4.05
	98.50
Total	434.0

Shuttle Cars	1		2	
Time	**Min**	**Pct**	**Min**	**Pct**
Production	275.10	63.39	243.85	56.19
Loading	93.65	21.58	76.00	17.51
Travel	110.55	25.47	104.40	24.06
Discharge	36.85	8.49	37.70	8.69
Place change	34.05	7.85	25.75	5.93

Delays—Shuttle Car No. 1	Min
Delay Time, 36.61 Pct	
Waiting on other car, 14.11 pct	61.25
Mechanical delays—loader, 9.52 pct	41.30
No empty mine car, 3.95 pct	17.15
Slate and lump handling (loader), 3.08 pct	13.35
Fall on loader, 1.87 pct	8.10
Handling supplies, 1.73 pct	7.50
Belt stoppages, 0.87 pct	3.80
Miscellaneous, 1.48 pct	6.45
	158.90

Delays—Shuttle Car No. 2	Min
Delay Time, 43.81 Pct	
Mechanical delays and supply handling, 14.86 pct	64.50
Waiting on other car, 12.80 pct	55.55
No empty mine cars, 5.52 pct	23.95
Mechanical delays—loader, 4.30 pct	18.65
Belt stoppage, 2.00 pct	8.70
Fall on loader, 1.96 pct	8.50
Slate and lump handling (loader), 1.58 pct	6.85
Miscellaneous, 0.79 pct	3.45
	190.15

Table III. Section C

Equipment: Crawler-mounted loading machine rated 5 tons per min, two cable-reel shuttle cars rated 4 tons, Crawler-mounted cutting machine (8½-ft bar, bottom cutting), Crawler-mounted drilling machine, 30-in belt conveyor, Elevating conveyor (belt feeder).

Personnel: Section foreman, Loading machine operator, Loading machine helper, Cutting machine operator, Cutting machine helper, Driller, Shot firer, two Shuttle car operators, Boom operator, Ventilation and water man, Rock duster, three Timbermen —**Total, 15.**

Operating Data: Mining height, 66 in.; Production, 343 tons; Number of shuttle car loads, 98; Average shuttle car capacity, 3.5 tons; Number of men, 15; Production per man-shift, 22.8 tons; Time in section, 385 min.

OPERATING CHARACTERISTICS

Loading Machine	Min
Production Time, 76.06 Pct	
Average loading time per car	1.37
Average change car time per car	1.21
Total per car	2.58
Average place change time per place	3.33
Average loading rate, tons	2.55
	292.80

Shuttle Cars	1	2
Trips	54	44
Travel speeds, ft per min	290	293
Average round trip, ft	720	720
Discharge time per trip, min	0.70	0.68
Discharge rate, tons per min	5.00	5.26

TIME ELEMENTS OF SHIFT STUDIED

Loading Machine	Min
Production Time, 76.06 Pct	
Loading time, 34.95 pct	134.55
Change car time, 30.73 pct	118.30
Place change, 10.38 pct	39.95
	292.80
Delay Time, 23.94 Pct	
Weld discharge elevator, 4.68 pct	18.00
No empty mine cars, 2.89 pct	11.15
Cable failure, shuttle car, 2.52 pct	9.70
Shoot adjoining place, 2.36 pct	9.10
Tramming delays, 1.77 pct	6.80
Ventilation, 1.50 pct	5.80
Slate and lump handling, 1.26 pct	4.85
Belt stoppages, 0.88 pct	3.40
Lubricate loader, 0.60 pct	2.30
Reset timber, 0.49 pct	1.90
Miscellaneous shuttle car, 2.52 pct	9.70
Miscellaneous—loader, 2.47 pct	9.50
	92.20
Total	385.00

Shuttle Cars	1		2	
Time	**Min**	**Pct**	**Min**	**Pct**
Production	283.95	73.75	194.20	50.44
Loading	76.00	19.74	58.70	15.25
Travel	138.60	36.00	103.00	9.35
Discharge	38.00	9.87	29.80	7.74
Place change	31.35	8.14	2.70	0.70

Delays—Shuttle Car No. 1	Min
Delay Time, 26.25 Pct	
Cable failures, 2, 4.52 pct	17.40
Waiting for other car, 4.31 pct	16.60
Weld discharge elevator, 4.14 pct	15.95
No empty mine cars, 2.83 pct	10.90
Shoot place, 2.37 pct	9.10
Belt stoppages, 2.15 pct	8.30
Slate and lump handling, 0.72 pct	2.75
Wait to travel, 0.66 pct	2.55
Reset timber, 0.48 pct	1.85
Handling supplies, 0.13 pct	0.50
Lubricate loader, 0.10 pct	0.40
Miscellaneous, 3.83 pct	14.75
	101.05

Delays—Shuttle Car No. 2	Min
Delay Time, 46.56 Pct	
Handling supplies, 25.51 pct	98.20
Waiting for other car, 7.56 pct	29.10
Weld discharge elevator, 4.87 pct	18.75
No empty mine cars, 3.41 pct	13.15
Other car cable failure, 2.64 pct	10.15
Belt stoppages, 1.32 pct	5.10
Ventilation, 1.08 pct	4.15
Slate and lump handling, 0.55 pct	2.10
Lubricate loader, 0.49 pct	1.90
Miscellaneous, 2.13 pct	8.20
	190.80

(Shuttle Cars) The largest delay was waiting at standby positions for the other car to clear, 14.11 pct and 12.80 pct. Handling supplies accounts for 1.73 pct and 14.86 pct, a portion of which was a controller delay. No empty mine cars accounted for 3.95 pct and 5.25 pct. Belt stoppages caused 2.00 pct and 0.87 pct. Miscellaneous delays were low at 1.48 pct and 0.79 pct. The remaining delays were directly accountable to loader stoppages. Two places had to be cleaned yielding only 1½ cars each. Proper cleanup initially would have prevented needless tramming time. The position of the shuttle car discharge point caused a change-car time at the belt also. Side loading can eliminate this possible source of delay. With established loading rate, discharge rate, and travel speed, change car time becomes a function of distance and standby points. Right of way, although universally given to the loaded cars, should be given to the empty cars in this case.

Section C

Fig. 3 is a layout of the trackless mining section using cable-reel shuttle cars. The seam mined averages 66 in. in thickness. The roof averages several inches of draw slate topped by hand slate; the bottom is hard fireclay. Mineability is considered relatively good. Considerable timbering is necessary. Five places 20 ft wide are driven on 50-ft centers. Crosscuts are angled at 60°.

Table III lists the equipment and personnel in the section.

ANALYSIS

Production Characteristics—The average loading rate for 29 loading machines of this type studied is 1.64 tons per min. This loader with an average of 2.55 tons per min is well above average. Place change is about average, long trams being balanced by very short ones. Travel distances of 720 ft per round trip are longer than the average of 680 ft for cable reel cars. Travel speeds of 290 ft per min are higher than an average of 275 ft per min for cable reel cars. Discharge rate at 5 tons per min is higher than average for cable reel car at 4.34 tons per min.

Delay Characteristics—(Loader) Mechanical delays accounted for 7.80 pct of the shift, the largest portion being a welding job on the elevating belt feeder. Poor coordination of section tramming and cable handling accounted for 1.77 pct of the face time. Lack of empty cars and belt stoppages caused by trip changes took 3.77 pct. Careless driving and miscellaneous shuttle car delays accounted for 3.01 pct. Miscellaneous delays to the loader was 2.47 pct, rather a high figure.

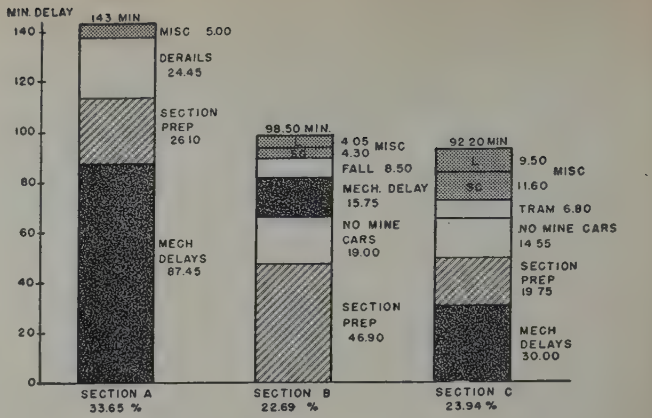

Fig. 5—Grouped loading delays in minutes.

Shuttle Cars: Handling supplies took 25.51 pct of the shift for one car, an abnormal amount. Wait at standby positions for other car was 4.31 pct and 7.56 pct. Lack of empty mine cars was 2.83 pct and halting shuttle car movement, 3.41 pct. Belt stoppages caused by changing mine car trips accounted for 2.16 pct and 5.10 pct. Two cable failures of one car took 4.52 pct while the other car was unable to travel for 2.64 pct. One car was delayed for 0.48 pct by careless driving or timbering that resulted in the need to replace a knocked out post under a header. Welding of the discharge elevating feeder was responsible for 4.14 pct and 4.87 pct. Miscellaneous delays were high at 3.83 pct and 2.13 pct, partially attributed to a new driver.

Supplies must be handled, and it is a characteristic of shuttle car belt sections that one shuttle car loses time to handle them. Evaluation of this time might indicate that a supply jeep or a timbering machine with trailer could be justified to eliminate the lost shuttle car time. Cable failures occurred at the discharge station, delaying both cars. Provision of two belt loading stations would have prevented the delay to one of the shuttle cars.

Section Comparisons: The relative performance of the loading machine with a shift average and a shift maximum is compared in Fig. 4. The delays according to groups and the actual time lost are given in Fig. 5. Fig. 6 shows the components of shift in percentage.

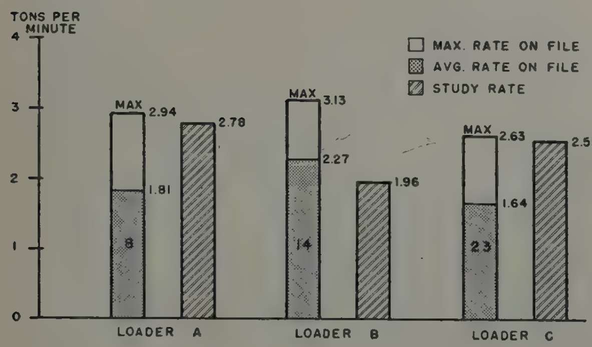

Fig. 4—Loading machine performance.

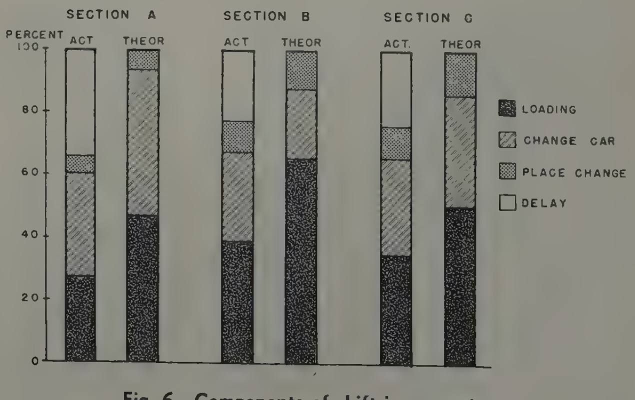

Fig. 6—Components of shift in percentage.

Theoretical Characteristics

For any given set of conditions it is desirable to know the maximum production possible. If this theoretical performance figure is developed, equipment ratings in one mine can be compared with another

It is obvious that a mathematical relationship exists between all the time elements making up an operation. For one shuttle car, the sum of travel empty, loading, travel loaded and discharging constitutes a productive cycle. This same principle can be applied to the loading machine using a single place as a basic operation. Equating the time elements for a single place to the productive effort:

Loading time per place + Change car time per place + Place change time = Productive effort per place.

The values for this equation are developed as follows: *Loading*—The average loading rate in minutes per car multiplied by the number of cars per place. *Change Car*—The average change car rate in minutes per car multiplied by the number of cars and modified from the average as follows:

One change car time is eliminated, assuming the car should be in position when the loader reaches the face in the new place.

A. The average used is for a full cut of nine cars including two trip changes.

B. Two cars are considered in operation for the full shift. Standby position at room neck is changed to the nearest breakthrough. The average is recalculated to obtain this.

C. Two cars are considered in operation for the full shift. The average is derived from the portion of the shift that they were in continuous operation.

Place Change—The average value for the study in minutes per place change.

Maximum Theoretical Production

Loading (min per car) × N (cars per place) + Change car (min per car) × (N-1) + Place change (min per place) = Productive effort (min per place)

Table IV. Theoretical Performance Rating

	Section A	Section B	Section C
Cars per cut	9	12	9
Loading time per car, min	2.22	1.79	1.37
Change car time per car, min	2.42	0.65	1.08
Place change avg, min	2.81	4.10	3.33
Cycle time per cut, min	42.15	32.73	24.30
Cuts per 420 min shift	9.96	12.83	17.28
Cars per shift	89.64	153.96	155.52
Tons per car	6.15	3.50	3.50
Tons per shift	551.3	538.9	544.3
Tons per manshift	39.4	49.0	36.3

From a time basis this is developed into production:

Places per shift × tons per place = tons per shift.

Table IV lists the theoretical performance possible for each of the sections analyzed for a common shift of 420 min.

The maximum theoretical production could be used as a standard; the product of careful study to determine the actual components of productive effort under existing conditions.

It should be emphasized that the preparation of the coal and the loading and tramming rate of the loading machines, governed partially by operating skill, greatly influence the production effort.

Production and Shift Ratings

From the theoretical calculations and obtained data it is possible to derive a production rating and a shift rating. The production rating is the comparison of the production with the theoretical characteristics and the actual production with the present characteristics. The shift rating is the comparison of the theoretical production for the full shift and the actual production for that shift.

Table V gives the calculations for these ratings while Figs. 7 to 10 are graphic comparisons.

The elimination of delays and increased production ratings cannot be emphasized too strongly.

Section A: The productive rating is 88.1 pct. Cleanup of several places yielding less than the average tonnage per cut contributes to this rating.

Table V. Production and Shift Ratings

	SECTION A		SECTION B		SECTION C	
	CHARACTERISTICS		CHARACTERISTICS		CHARACTERISTICS	
	PRESENT	THEOR.	PRESENT	THEOR.	PRESENT	THEOR.
Existing Full Shift	491.3	557.9	429.4	556.9	450.9	498.9
Effort, pct	66.35		77.31		76.06	
Production for present effort	326.	370.1	332.	430.5	343.	379.5
Productive rating, pct	$\frac{326.}{370.1}$ =	88.08	$\frac{332.}{430.5}$ =	77.12	$\frac{343.}{379.5}$ =	90.38
Shift rating, pct	$\frac{326.}{557.9}$ =	58.44	$\frac{332.}{556.9}$ =	59.62	$\frac{343.}{498.9}$ =	68.75
Efficiency loss, pct	11.92		22.88		9.62	
Delay loss, pct	$\frac{491.3-326}{557.9}$ =	29.64	$\frac{429.4-332}{556.9}$ =	17.50	$\frac{450.9-343}{498.9}$ =	21.63

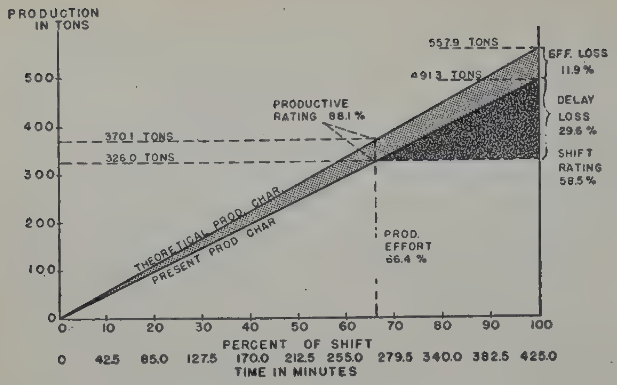

Fig. 7—Section A production characteristics.

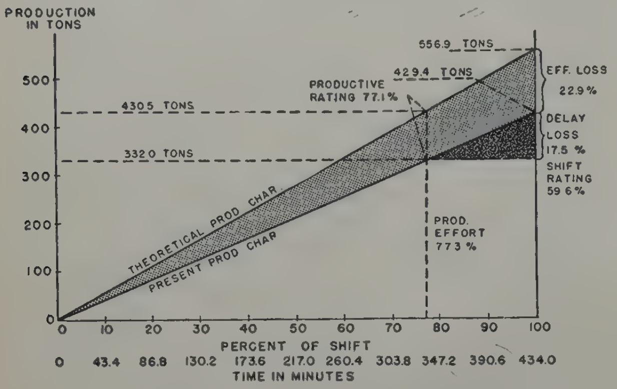

Fig. 8—Section B production characteristics.

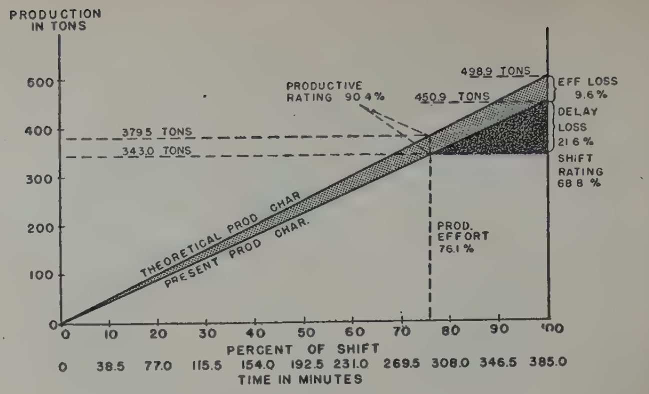

Fig. 9—Section C production characteristics.

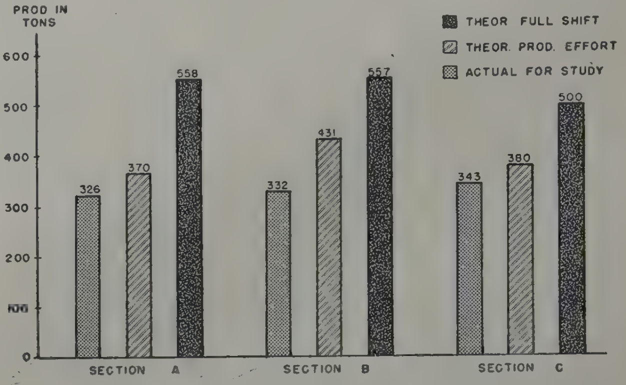

Fig. 10—Comparison of theoretical and actual production.

The shift rating is 58.5 pct with efficiency loss accounting for 11.9 pct and delay loss accounting for 29.6 pct.

Section B: The productive rating is 77.1 pct. Breakdown of one car or handling supplies by one car and the inefficient standby position in addition to revisiting cuts for cleanup accounts for this low percentage. The shift rating is 59.6 pct with efficiency loss accounting for 22.9 pct and delay loss accounting for 17.5 pct.

Section C: The production rating is 90.4 pct. Handling supplies and breakdown of one car not affecting the operation of the other and cleaning of partially loaded places account for this rating. The shift rating is 68.8 pct with efficiency loss accounting for 9.6 pct and delay loss accounting for 21.6 pct.

Continuous Mining

The more important deficiencies encountered in the use of continuous mining machines are well known, but these are being corrected and improved. The present machines produce a product that consists of a large proportion of small sizes and, as in operations where spraying is mandatory, the moisture content of the product is higher. The small sizes place an additional load on the cleaning plant, and where moisture and size must meet rigid specifications, the problem becomes more difficult.

Costly capital expenditures and changes of market for the product must be carefully considered before entering into continuous mining. With conventional mining systems, minor changes in face preparation can often be made to alter the product to meet market specifications. With continuous mining systems, flexibility of product is limited.

Fig. 11 shows the labor productive comparison of conventional mining sections and continuous mining sections in the same mines operating in the central Pennsylvania area.

The information in Fig. 11 can be misleading if all factors are not considered. The continuous mining machines are restricted by seam conditions, and the increased labor productivity is not always realized as profit because of the change in realization from the marketed product.

It is believed that conventional mechanized mining systems will be employed successfully for a considerable time. Modernization of mines can be done efficiently without converting to continuous mining machines. However, the mine operator should look forward to the day when he may want to convert his operation to continuous mining and should consider major capital expenditures and improvements with the thought of their future relationship with continuous mining.

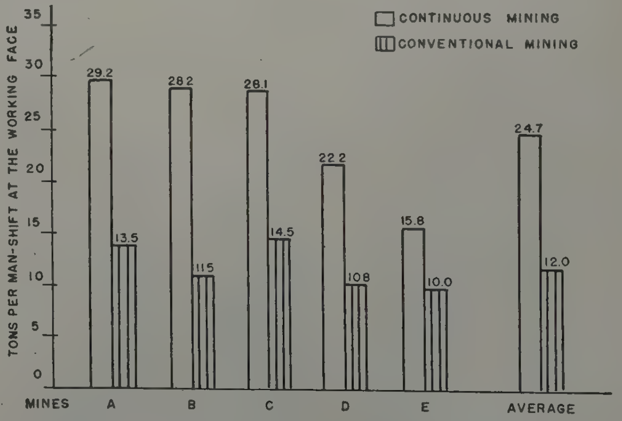

Fig. 11—Performance comparison of equipment working in the same mines, low volatile bituminous coal in central Pennsylvania.

Destruction of Flotation Froth With Intense High-Frequency Sound

by Shiou-Chuan Sun

THE presence of an excessive amount of tough froth in the flotation of minerals, particularly coals, may create trouble in dewatering, filtering, and handling. Froth is also a nuisance in many chemical industries.[1] This paper presents a study on the destruction of extremely tough froths with intense high-frequency sound. The data indicate that sound waves can be employed for continuous and instantaneous defrothing. A powerful high-frequency siren was used in obtaining the data.

Also tested was an ultrasonorator of the crystal type with a frequency range of 400, 700, 1000, and 1500 kc per sec and a maximum power output from its amplifier of 198 w. The results, not presented, indicate that as now designed this machine is not suitable for defrothing. Although the sound generators of the magnetostriction type[2,3] and of the electromagnetic type[4,5] were not available, it is believed they are capable of producing the required sound intensity for defrothing.

The use of ultrasonics for defrothing was suggested by Ross and McBain[1] in 1944. Ramsey[6] reported in 1948 that E. H. Rose mentioned a supersonic device that broke down flotation froth but with low capacity. The writer has not been able to find any published literature containing practical experiments.

Theoretical Considerations

The mechanism of defrothing by sound is attributed to the periodically collapsing force of the propagated sound waves and the induced resonant vibration of the bubbles. The collapse of froth is further facilitated by the sonic wind and the heat of the siren.

Sound waves can exert a radiation pressure[7,8] against any obstacle upon which they impinge. When a froth surface is subjected to the periodic puncturing of sound waves, the bubbles are broken. According to Rayleigh[9] and Bergmann,[12] the radiation pressure of sound, P, in dynes per sq cm is given as:

$$P = \tfrac{1}{2}\,(r+1)\,\frac{i}{v}$$

where r is the ratio of the specific heats of the medium through which sound is traveling and is equal to 1 on the basis of Boyle's law; i is the sound intensity in ergs per sec per sq cm, and v is the sound velocity in cm per sec. In this case, the accuracy of the formula is only approximate, because a perfect reflection can hardly result from a column of froth.

In addition to the radiation pressure, the propagated sound waves cause the bubbles of the froth to have a resonant vibration.[10] The vibratory motion of the bubbles causes collision and coalescence, thereby weakening if not breaking the bubble walls. Sonic wind and heat were also generated.[11] The sonic wind can exert pressure on the froth surface, and the heat can evaporate the moisture content of the bubble walls as well as expand the enclosed air.

Apparatus

The defrothing apparatus, shown in Figs. 1 and 2, consists of a powerful high-frequency siren, a glass or stainless steel beaker of 2-liter capacity with 12.4 cm diam and 17.1 cm height, and a metal reflector. The beaker was placed 2 in. above the top point of the siren. The metal reflector was adjusted to reflect and focus the generated sound waves into the central part of the beaker. Fig. 2 shows the crystal probe microphone used to measure the acoustic intensity and the mandler bacteriological filter employed to introduce compressed air into the beaker for frothing. The apparatus was enclosed in a soundproof cabinet equipped with a glass window.

The siren, shown in Fig. 3, consists of a rotor that interrupts the flow of air through the orifices in a stator. The rotor, a 6-in. diam disk with 100 equally spaced slots, is driven by a 2/3 hp, Dumore W2 motor at 133 rps. The frequency of the siren can be varied from 3 to 34 kc. The maximum chamber pressure is about 2 atm, yielding acoustic outputs of approximately 2 kw at an efficiency of about 20 pct. The siren itself is relatively small and can be operated in any orientation. A detailed description of the siren has been given by Allen and Rudnick.[11]

Collapse of Froth

To study the sequence of the collapse of froth, the glass beaker was partially filled with 920 cc water, 100 g of —150 mesh bituminous coal, 0.3 cc petroleum light oil, 0.2 cc pine oil and 1.54 cc Pyrene foam compound. This mineral pulp was agitated for 5 min and then aerated through a mandler filter until the empty space of the beaker, approximately 9 cm high, was filled completely with min-

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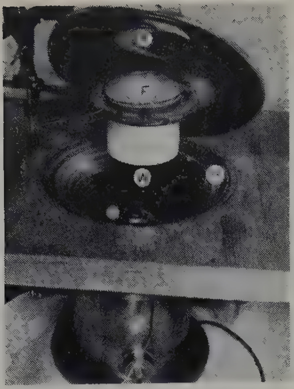

Fig. 1—Defrothing apparatus.

R—Metal reflector
W—Liquid or mineral pulp
F—Froth column
H—Horn of siren
S—Siren

Note that the froth was purposely prepared without the presence of mineral particles for the sake of better visualization in the picture.

eralized froth. The siren was set at a frequency of 7 kc per sec, and a chamber pressure of 30 psi was maintained. The sequence of the collapse of froth was photographed with a Cine-Kodak Special, 16 mm motion picture camera at a speed of 16 frames per sec. The result of this study is shown in Fig. 4.

In Fig. 4, frame 0 shows that the froth was highly mineralized (in air the froth could persist for 4 hr). Frame 4 shows that when the froth column was exposed to the sound waves for ¼ sec, a round hole approximately 2 cm diam and 2 cm deep was punctured in the top. Frames 9 and 18 show that, after ½ sec and more, the bubbles vibrated violently. The longitudinal bubble waves with crest and troughs suggest the presence of resonant vibrations. Frames 26 and 36 show broken fragments, chiefly coal particles and water droplets, being dissipated into air at a maximum height of approximately 2 in. This may be caused by the force of 1—the sound waves reflected back from the froth, 2—the released air pressure from the broken bubbles, and 3—the vibratory motion of the bubbles.[10] The vertical black lines and dark color in the liquid phase of frames 48 and 61 indicates that the froth column was completely destroyed and the once floated coal particles fell back into the liquid. Approximately 4 sec were required to collapse the froth column.

Rate of Defrothing

A highly mineralized froth, prepared by the same procedure, was tested with different frequencies at a chamber pressure of 30 psi. The results, as plotted in curve 1 of Fig. 5, show that the rate of defrothing increases with the decrease of the frequency up to 6 kc per sec. This may be because the power output of the siren within certain limits is increased by the decrease of frequency, as shown in curve 2 of Fig. 5. The data of curve 2 were established by Allen and Rudnick[11] at a constant chamber pressure of 2.94 psi.

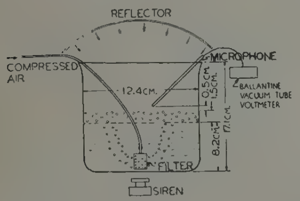

Fig. 2—Schematic diagram of the defrothing apparatus.

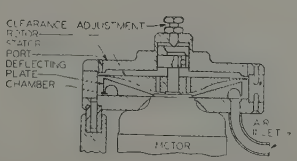

Fig. 3—Cross-section of the siren.

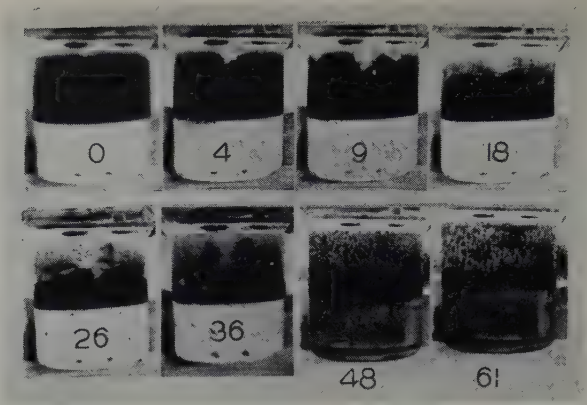

Fig. 4—Movie photographs showing the sequence of collapse of a highly mineralized froth column.

Note that the mineralized froth occurred only at the very black portion of the photographs; above that the less black portion was caused by the dissipation of the broken bubble fragments in air or by the attachment of these fragments on the inner surface of the glass beaker.

Froth Elimination

The prevention of froth on liquid surface by intense high-frequency sound was tested and proved satisfactory. The apparatus was set up as shown in Fig. 2; the beaker was filled with 920 cc water, 0.3 cc petroleum light oil, 0.2 cc pine oil, and 1.54 cc Pyrene foam compound and was agitated for 5 min. Compressed air was forced through a mandler filter, which was kept in the liquid during the entire testing period. One second after the mandler filter was introduced into the beaker, the chamber pressure and the frequency of the siren were varied simultaneously to form different combinations for the elimination of froth on the liquid surface. The results of the test are given in Table I.

Table I indicates that frothing can be prevented by bombarding the pulp surface with suitable sound waves. For the generation of a constant sound intensity, the relationship existing between the chamber pressure and the frequency of the siren is a straight line, as shown in Fig. 6.

Mathematically, this can be expressed as:

$$y = k + mx$$

Where y is the ordinate representing the chamber pressure in psi, k is the ordinate intercept at 0 abscissa, x is abscissa representing the sound frequency in kc per sec, and m is the slope of the curve.

Industrial Defrothing

The noise of the siren would be objectionable in its industrial application for defrothing. This could

Table I. The Prevention of Frothing on Liquid Surface

Test No.	Frothability of Solution with a Mandler Filter, Cm per Sec	Observed Height of Froth, Cm	Siren			
			Chamber Pressure, Psi	Frequency, Kc per Sec	Sound Intensity	
					W per Cm^2	Ballantine Decibels[a]
1	0.07	0.07	none	none	none	none
2	0.07	none	11	6.0	0.05	24
3	0.07	none	22	8.8	0.06	25
4	0.07	none	25	10.3	0.06	25

[a] Decibels relative to 1 mv.

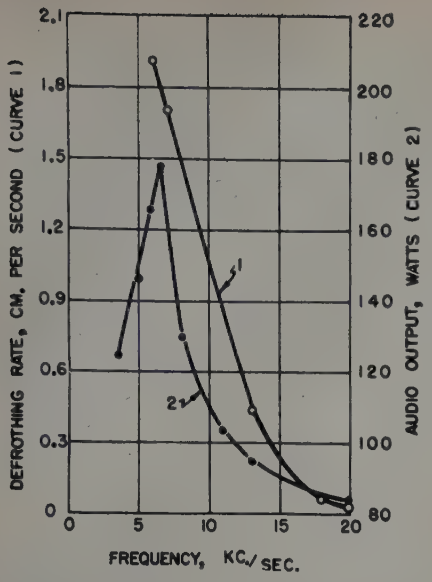

Fig. 5 — Effect of sound frequency on the rate of defrothing and on the power output of the siren.

be remedied by enclosing both the siren and the upper part of the container in a soundproof cabinet, as shown in Fig. 7. The walls and ceiling of the cabinet can be constructed with wood or steel. Based on the experience of laboratory operation of the siren, the inner wall surfaces of the cabinet should be lined with a 6-in. thickness of cotton or other suitable sound-absorbing materials. At one side of the cabinet a door with one or two glass windows is provided for observation and accessibility.

Fig. 7 shows a few possible arrangements for industrial defrothing. The siren should be supported so that its elevation and horizontal position can be adjusted easily. Diagram 1 of Fig. 7 is a cylindrical container suitable only for intermittent defrothing of small-scale operations. Diagram 2 shows a rectangular or cylindrical container, capable of continuous defrothing of a medium size operation. Diagram 3 shows a launder for continuous defrothing.

Summary

1—A tough froth column can be destroyed instantaneously by means of a powerful high-frequency siren.

2—The surface of a highly frothable liquid can be kept free of froth by bombarding it with suitable sound waves.

3—For a given chamber pressure of 30 psi, the optimum frequency of defrothing for the particular siren used is about 6 kc per sec.

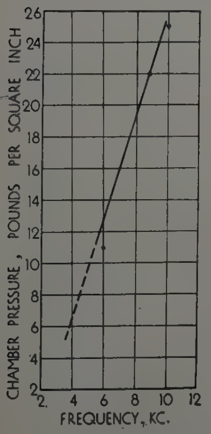

Fig. 6 — Relationship between chamber pressure and frequency for the siren to produce a constant sound intensity.

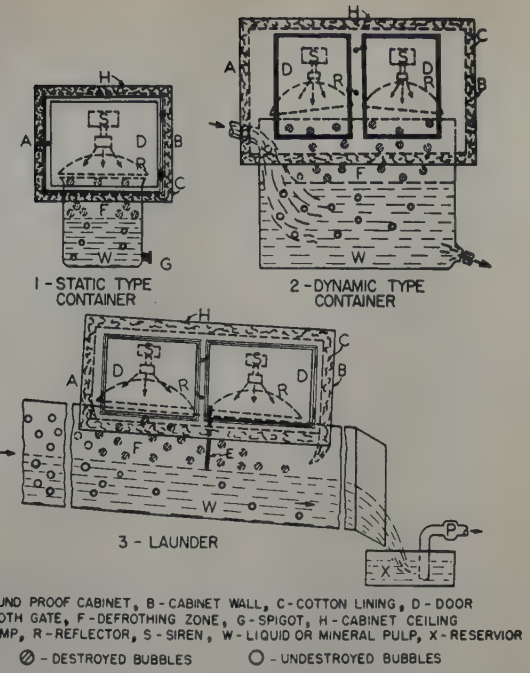
1 - STATIC TYPE CONTAINER
2 - DYNAMIC TYPE CONTAINER
3 - LAUNDER
A
B
C
D
E
F
G
H
P
R
S
W
X
A - SOUND PROOF CABINET, B - CABINET WALL, C - COTTON LINING, D - DOOR
E - FROTH GATE, F - DEFROTHING ZONE, G - SPIGOT, H - CABINET CEILING
P - PUMP, R - REFLECTOR, S - SIREN, W - LIQUID OR MINERAL PULP, X - RESERVIOR
⊘ - DESTROYED BUBBLES
○ - UNDESTROYED BUBBLES

Fig. 7—Some of the possible arrangements for industrial defrothing.

Acknowledgments

The writer wishes to express his thanks to Dr. C. H. Allen for his kind assistance and invaluable advice on the operation of the siren. He is also indebted to Professor D. R. Mitchell for his encouragement and administrative guidance.

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Filter Cake Size-Consist And Moisture Relationships

by Orville R. Lyons

THE continuous vacuum filter, whether it be a vertical or horizontal disk, inside or outside drum, when properly applied is a very effective device for dewatering fine solids. Because of the number of variables that affect its performance, however, it is not possible to obtain consistently maximum moisture reduction in an operating plant. This paper presents a method for determining the average cake moisture and the variations in cake moisture to be anticipated when dewatering fine solids by continuous vacuum filters under normal operating conditions.

Previous investigators[1, 2] have found that certain factors influence filter performance, drum or disk diameter, speed of rotation, percentage of immersion, and the percentage of solids in the feed influence capacity. Blinding of the filter cloth reduces effective vacuum, produces a wetter cake, and may decrease capacity. Reduced vacuum or improper adjustment between vacuum and blowing portions of the cycle will produce a wetter cake and also may decrease capacity. Widely fluctuating feed tonnages will adversely affect cake moisture and capacity. Flocculation, when properly applied, increases capacity and reduces cake moisture. Without flocculation, the slimes are the first particles sucked to the filter medium, which becomes partially blinded in consequence. With flocculation, the flocs act as coarse particles and the filter medium is not blinded. Overflocculated pulps, although filtering well, may yield wet cakes.

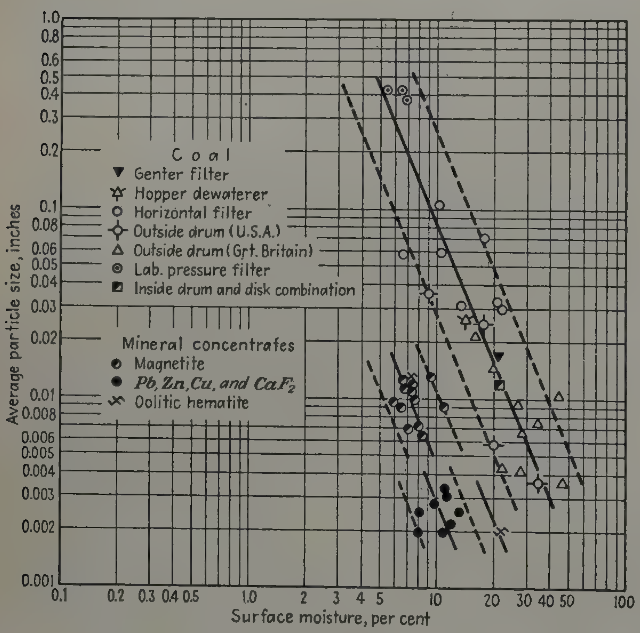

Fig. 1—Relationship between average particle size and moisture content of cake when dewatering coal and mineral concentrates by means of filters.

Average Particle Size and Cake Moisture

A method for predetermining the surface moisture content of coal filter cakes was published in 1949,[3] but this method could not be applied to coals deficient in —200 mesh material or to fine mineral concentrates.

Test data obtained for geared-weight types of vibrating screens[4] and for centrifuges[5] used to dewater coal provided definite correlations between the average particle size of a dewatered cake and the surface moisture content of the cake. It was thought that this same method of correlation should be applicable to filters, whether dewatering fine coal or fine mineral concentrates.

To develop a correlation or correlations between average size of cake and cake moisture, the author collected data for filtration operations from as many sources as possible. In a considerable number of instances the information was provided by plant preparation engineers,[6-8] a few of the tests were on a laboratory scale,[9] and the remainder of the data was obtained from published articles.[10-17]

Table I contains size-consist and miscellaneous data for coal filter cakes, and Table II contains the same data for mineral concentrate filter cakes. Fig. 1 shows the relationships existing between the average particle size of the cake and cake moisture (surface moisture for coal and total moisture for minerals) when dewatering fine solids by a variety of makes and types of filters.

Average particle size is calculated by averaging the linear openings of the various pairs of screens used in the screening test, weighted in accordance with the percentages of material in the various fractions.

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Table I. Size-Consist and Miscellaneous Data for Various Coal Filter Cakes

Mesh Size	Test No. 1 Wt, Pct	Test No. 2 Wt, Pct	Test No. 3 Wt, Pct	Test No. 4 Wt, Pct	Test No. 5 Wt, Pct	Test No. 6 Wt, Pct	Test No. 7 Wt, Pct	Mesh Size	Test No. 8 Wt, Pct	Test No. 9 Wt, Pct
4	0.6		0.4							
4 x 6	4.3	1.1	1.6	0.9	11.2	1.7	21.7	+48	6.0	2.0
6 x 8	17.2	6.0	6.8	6.1	11.0	3.9	14.7	48 x 100	31.0	7.5
8 x 10	16.0	8.6	8.7	8.9	11.4	14.1	14.5	100 x 200	29.0	15.0
10 x 20	25.8	19.8	12.8	16.2	9.8	15.3	10.7	−200	34.0	75.3
20 x 28	9.8	10.2	7.7	9.9	13.0	22.4	13.5			
28 x 48	13.4	12.4	15.6	15.4	5.2	8.9	4.6			
48 x 65	4.8	4.8	7.7	5.5	7.5	16.4	9.7			
65 x 100	3.4	6.7	7.7	6.7	3.8	7.5	4.4			
100 x 200	2.3	10.2	10.7	9.6	4.2	5.1	3.1			
−200	2.4	20.2	20.3	19.8	6.8	2.5	1.7			
					16.1	2.2	1.4			
Average cake size, in.	0.0586	0.0319	0.0314	0.0307	0.0729	0.0584	0.1054		0.0057	0.0038
Surface moisture of cake, pct	6.5	13.6	20.5	21.1	17.5	10.2	10.3		20.3	35.2
Type of filter	[a]	[a]	[a]	[a]	[illegible]	[a]	[a]		[b]	[b]
Coal seam	[c]	[c]	[c]	[c]	[c]	[c]	[c]		[d]	[d]

Mesh Size	Test No. 10 Wt, Pct	Test No. 11 Wt, Pct	Test No. 12 Wt, Pct	Test No. 13 Wt, Pct	Test No. 14 Wt, Pct	Test No. 15 Wt, Pct	Test No. 16 Wt, Pct	Test No. 17 Wt, Pct	Test No. 18 Wt, Pct
1/16 in. x 32	40.2	23.6		9.8	0.6	5.8	4.5	12.5	
32 x 60	19.3	16.1	5.7	19.5	1.7	5.6	17.6	16.6	1.8
60 x 100	22.0	15.2	8.6	9.6	4.9	8.6	9.6	12.6	5.4
100 x 200	12.5	21.6	27.2	11.8	29.1	19.6	14.8	20.8	14.4
−200	7.0	23.5	58.5	49.2	63.7	60.4	53.5	37.5	78.4
Average cake size, in.	0.0219	0.0150	0.0045	0.0096	0.0042	0.0066	0.0075	0.0107	0.0037
Surface moisture of cake, pct	16.0	19.8	23.0	26.9	28.0	28.7	34.3	45.7	47.8
Type of filter	[b]	[b]	[b]	[b]	[b]	[b]	[b]	[b]	[b]
Coal seam	[e]	[e]	[e]	[e]	[e]	[e]	[e]	[e]	[e]

Mesh Size	Test No. 19 Wt, Pct	Mesh Size	Test No. 20 Wt, Pct	Test No. 21 Wt, Pct	Mesh Size	Test No. 22 Wt, Pct	Test No. 23 Wt, Pct	Mesh Size	Test No. 24 Wt, Pct
10 x 14	19.0	8 x 14	8.3	29.1	¾ in. x ½ in.	26.9	26.9	¾ in. x ½ in.	24.8
14 x 28	32.5	14 x 28	33.4	30.1	½ in. x 3	53.7	53.7	½ in x 3	49.7
28 x 32	5.5	28 x 48	30.3	19.9	3 x 6	9.7	9.7	3 x 6	9.0
32 x 60	17.0	48 x 100	15.4	12.4	6 x 10	9.7	9.7	6 x 10	9.0
60 x 115	11.8	100 x 200	7.4	4.5				10 x 28	5.0
115 x 250	5.7	−200	5.2	4.0				28 x 35	2.5
−250	8.5								
Average cake size, in.	0.0267		0.0244	0.0355		0.4016	0.4016		0.3744
Surface moisture of cake, pct	14.5		17.0	9.0		6.46	5.37		6.84
Type of filter	[f]		[b]	[b]		[g]	[g]		[g]
Coal seam	[h]		[i]	[h]		[j]	[j]		[j]

Mesh Size	Test No. 25 Wt, Pct	Mesh Size	Test No. 26 Wt, Pct
+20	1.17	14 x 20	13.10
20 x 30	5.49	20 x 48	32.75
30 x 40	13.85	48 x 65	11.79
40 x 50	10.67	65 x 100	5.24
50 x 60	8.75	100 x 200	7.96
60 x 70	3.88	200 x 325	6.45
70 x 80	5.23	−325	22.71
80 x 100	18.60		
100 x 140	24.96		
140 x 200	4.57		
−200	2.83		
Average cake size, in.	0.0120		0.0162
Surface moisture of cake, pct	21.5		21.0
Type of filter	[k]		[l]
Coal seam	[d]		[d]

[a] Horizontal filter
[b] Outside drum filter
[c] Miller B and Upper Freeport (Mixed)
[d] Pennsylvania, Pittsburgh
[e] Great Britain, various seams
[f] Hopper dewaterer
[g] Laboratory pressure filter
[h] Ohio, Pittsburgh No. 8
[i] West Virginia, No. 2 Gas
[j] Illinois No. 5
[k] Inside drum and disk combination
[l] Genter filter

Fig. 1 reveals that coal has one size-moisture relationship and that mineral concentrates have other but parallel relationships. Since most of the data were gathered under operating conditions, the dotted lines enclosing 1—the coal points and 2—the mineral points represent the extremes in cake moisture content that would be obtained in plant operation.

These sets of relationships are not identical because of 1—specific gravity differences and 2—shape and surface characteristic differences. Coal particles, whether hard and blocky or platy and friable apparently have very similar shape and surface characteristics. Mineral concentrates fall into three groups, granular concentrates, flotation concentrates, and slime concentrates. The solid lines, Fig. 1, represent the average conditions that may be obtained in an operating plant. One such average condition is shown for coal, one for magnetic and oolitic granular iron concentrates, and one for sulphide and fluorspar flotation concentrates. Another such average condition could be anticipated for the slime iron concentrate (Test 21). It seems indicated that these average values should be used for the various types of materials when predicting operating results.

The wide variations in cake moisture obtained under operating conditions, when treating a given size-consist material, are the result of 1—fluctuations in plant tonnage, 2—fluctuations in feed solid content, 3—gradual wearing or blinding of filter cloths, and 4—the approximate method used to determine surface area.

Apparently there is a minimum moisture content obtainable for any given size-consist of cake. When applying the dewatering forces that can be brought to bear on the particles, using makes of filters currently available, the water film remaining on the particles has a minimum thickness for any particular material and therefore minimum cake moisture is a direct function of surface area. Failure to operate

Table II. Size-Consist and Miscellaneous Data for Various Mineral Concentrate Filter Cakes

Mesh Size	Test No. 1 Wt, Pct	Test No. 2 Wt, Pct	Test No. 3 Wt, Pct	Test No. 4 Wt, Pct	Test No. 5 Wt, Pct	Mesh Size	Test No. 6 Wt, Pct	Test No. 7 Wt, Pct	Test No. 8 Wt, Pct	Test No. 9 Wt, Pct	Test No. 10 Wt, Pct
+10	0.23	0.22	1.1	0.5	0.37	+20	1.4	2.8	2.3	1.4	1.17
10 x 20	0.84	0.75	1.9	1.8	1.87	20 x 100	47.5	52.3	41.8	50.3	49.23
20 x 35	5.03	4.93	9.4	11.2	10.15	100 x 200	23.0	22.3	26.3	28.2	23.36
35 x 65	20.8	22.8	27.8	24.3	30.55	−200	28.1	22.6	29.6	20.1	26.24
65 x 100	17.3	17.4	16.1	16.1	13.86						
100 x 200	26.9	30.2	21.4	17.6	21.50						
−200	28.9	23.7	22.3	28.5	21.70						
Average cake size, in.	0.0069	0.0071	0.0093	0.0092	0.0096		0.0113	0.0127	0.0108	0.0119	0.0115
Total moisture of cake, pct	7.0	6.0	6.3	10.9	6.0		7.3	9.2	7.1	6.7	7.2
Type of filter	a	a	a	a	a		a	a	a	a	a
Material filtered	b	b	b	b	b		b	b	b	b	b

Mesh Size	Test No. 11 Wt, Pct	Mesh Size	Test No. 12 Wt, Pct	Mesh Size	Test No. 13 Wt, Pct	Test No. 14 Wt, Pct	Mesh Size	Test No. 15 Wt, Pct	Test No. 16 Wt, Pct	Mesh Size	Test No. 17 Wt, Pct
20 x 28	2.85	+100	7.78	+48	0.6						
28 x 35	3.86	100 x 150	9.27	48 x 65	2.3	0.8	+150	0.6	2.0	+100	6.5
35 x 48	8.50	150 x 200	6.96	65 x 100	7.9	2.4	150 x 200	3.8	7.0	100 x 150	12.5
48 x 65	12.58	200 x 325	12.19	100 x 150	14.0	5.1	200 x 325	15.8	24.4	150 x 200	15.2
65 x 100	15.48	−325	63.80	150 x 200	16.6	10.7	−325	79.8	66.6	−200	65.8
100 x 150	11.63			200 x 270	13.7	14.7					
150 x 200	13.58			−270	44.9	66.3					
−200	31.52										
Average cake size, in.	0.0069		0.0026		0.0034	0.0026		0.0018	0.0020		0.0031
Total moisture of cake, pct	8.0		13.0		11.0	8.0		8.0	11.0		11.0
Type of filter	a		c		c	c		c	d		c
Material filtered	b		e		f	g		h	f		i

Mesh Size	Test No. 18 Wt, Pct	Mesh Size	Test No. 19 Wt, Pct	Mesh Size	Test No. 20 Wt, Pct	Test No. 21 Wt, Pct
+ 100	5.0	+150	4.8	+14	0.8	
100 x 200	14.0	150 x 200	15.5	14 x 20	2.5	
−200	81.0	200 x 325	19.6	20 x 35	15.7	
		−325	60.1	35 x 65	40.3	0.1
				65 x 100	25.1	1.5
				100 x 200	12.9	10.1
				−200	2.7	
				200 x 325		14.3
				−325		74.0
Average cake size, in.	0.0028		0.0022		0.0126	0.0021
Total moisture of cake, pct	10.0		12.0		7.0	22.0
Type of filter	c		c		c	d
Material filtered	i		i		j	k

[a] Inside drum filter
[b] Magnetic iron concentrates
[c] Outside drum filter
[d] Disk filter
[e] Copper concentrate
[f] Zinc concentrate
[g] Lead concentrate
[h] Copper-lead concentrate
[i] Fluorspar concentrate
[j] Oolitic iron concentrates
[k] Slime iron concentrate

a filter properly will provide a cake having a higher than theoretical moisture.

Conclusions

By means of the relationships presented, it is possible to predetermine average cake moisture and the variations in cake moisture to be anticipated when dewatering coal, granular mineral concentrates, flotation mineral concentrates, and slime mineral concentrates by vacuum filters under normal operating conditions. Apparently all of the makes of filters currently available are equally effective from a moisture reduction standpoint.

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Using Electric Furnaces and Heaters To Determine the Free-Swelling Index of Coal

by E. Swartzman and G. C. Behnke

THE free-swelling index, a numerical value for the free-swelling properties of coal, is being used to an increasing extent in specifying coal for burning in various types of equipment. The empirical standard ASTM method for determining this index uses, as a source of heat, a gas burner which requires frequent calibration because gas pressures are not constant, relatively small changes in gas pressure resulting in rather wide variations in the temperature attained in a crucible and thus in the resultant size of the coke *button*. Furthermore, the quartz crucibles specified for the test are not standard equipment, and relatively small variations in dimension and wall thickness from those specified often result in rather wide variations in the resultant size of the button.

A method of analysis to be satisfactory not only for a scientific laboratory but also for use in industrial control laboratories under varying conditions should employ equipment which is rugged, easily calibrated, and readily replaceable by standard materials. Gas is not universally available as is electricity, and it is generally easier to calibrate and control a simple electric furnace or heater than a gas burner, the voltage not usually fluctuating to the degree that gas pressure does. Quartz crucibles are fragile, and the size specified in the test is not normally used for any other purpose in the laboratory. Although more expensive, the 15-cc platinum crucible, available in all laboratories where volatile matter in coal is determined by the standard ASTM method, is sturdy and standard and easily replaced.

Because of these considerations, a modification of the standard method has been developed which employs a standard laboratory 350-w cone heater and standard 15-cc platinum crucibles. The results of numerous tests conducted with such apparatus show conclusively that the free-swelling index values as judged by the buttons produced with this equipment are exactly the same as those obtained by the standard gas burner method.

Thus it is suggested that this electric cone heater method employing standard 15-cc platinum crucibles be considered as an alternative to the ASTM method, Designation: D720-46.

The free-swelling index of coal is determined by means of the ASTM method, D720-46. This method, which is an adaptation of the crucible swelling test for coal of the British Standards Institution, consists of heating 1 g of coal in a special covered quartz crucible in a regulated gas flame for a period of not less than 2.5 min in which time the temperature of the inner surface of the bottom of the crucible reaches a temperature of 820±5°C. The residual coke button is then compared in size and shape with a series of standard profiles shown in Fig. 1 and designated by number from 1 to 9 in increasing order of size.

As the method is empirical, it requires rigid standardization of the apparatus and repeated calibration of the gas burner. The dimensions of the low form quartz crucible recommended must be rigidly adhered to as small variations in diameter or wall thickness result in variations in the rate of heat transmission and thus in the resultant size and shape of the coke button. The gas burner must be calibrated almost every time a new set of determina-

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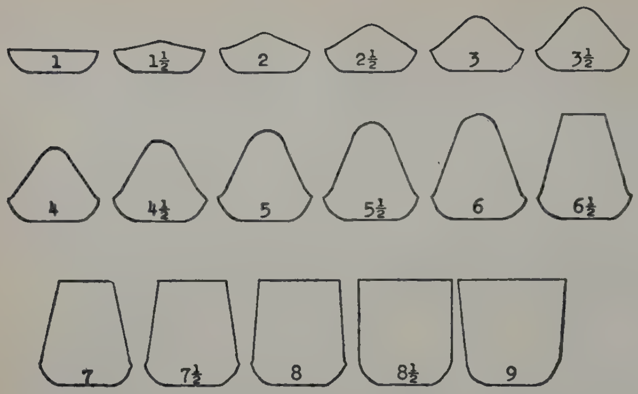

Fig. 1—Standard profile and corresponding swelling index numbers. Reproduced approximately one half full size.

tions are made because gas pressures are not constant. In addition, the method is not universally adaptable because gas is not available at all laboratories, especially at mines where it may be desirable to check every car of coal prior to shipment. Furthermore, it requires the use of equipment which is not standard and not always readily available especially insofar as the crucibles are concerned.

A method which is to be satisfactory for use at a mine tipple, coke plant, railway yard, power plant, or other industrial plant should be fairly rugged, easy to calibrate, simple to operate, and make use of standard equipment readily obtainable.

With this in view, a method has been developed whereby a simple electric laboratory cone heater supplies the source of heat and the standard covered 15-cc platinum crucible, used in determining the volatile matter of coal by the standard ASTM method is employed for preparing the coke buttons, which are then compared to the standard profiles. The coal is heated at a temperature of 650°C, inside the crucible bottom, for 4 min.

Equipment and Coals Tested

Two types of electric heaters were used in this investigation.

The first experiments were conducted with a 485-w multiple unit electric vertical tube furnace with an inside depth of 4½ in. and inside diam of 2⅜ in. Temperature control was attained by controlling the voltage with a Variac.

A 350-w electric cone heater, recommended as a substitute for the Bunsen burner, was used in the second group of experiments. This heating unit consists of an open helical spiral element of chromel wire lightly secured by alundum cement to the sides of a conical depression in a refractory heat-insulating block. The conical arrangement of the glowing wire focuses the radiant heat to produce the so-called spot-heating effect of a Bunsen burner.

This heater has an overall height of 6¼ in., and the top, 5¾ in. diam, is fitted with three concentric rings with openings of 1¼ in., 2 in., and 3⅛ in. diam. Depending upon the size of the crucible, rings of varying diameter had to be made up, so that the crucible fits well into the furnace with only about ⅛ in. protruding above the ring. To prevent undue loss of heat due to convection, a transite chimney 1¾ in. high and about 4 in. in diam was placed on the top. Temperature control was attained by means of a Variac. Figs. 2 and 3 show the details of the cone heater.

Table I. Description of Canadian Coals

Sample No.	Origin	Seam	Size	Rank[a]
B13612	Sydney Area, Nova Scotia	Harbour	Channel	H.V.A. Bit.
B13666	Sydney Area, Nova Scotia	Harbour	Channel	H.V.A. Bit.
B13735	Sydney Area, Nova Scotia	Phalen	Slack	H.V.A. Bit.
32625	Sydney Area, Nova Scotia	Mixed	Slack	H.V.A. Bit.
32649	Sydney Area, Nova Scotia	Mixed	Slack	H.V.A. Bit.
B13702	Sydney Mines Area, Nova Scotia		Lump	H.V.A. Bit.
B13704	Sydney Mines Area, Nova Scotia		Lump	H.V.A. Bit.
32736	Sydney Mines Area, Nova Scotia	Harbour	Stoker	H.V.A. Bit.
32920	Sydney Mines Area, Nova Scotia	Harbour	Channel	H.V.A. Bit.
32940	Sydney Mines Area, Nova Scotia	Harbour	Channel	H.V.A. Bit.
32958	Sydney Mines Area, Nova Scotia	Harbour	Channel	H.V.A. Bit.
32976	Sydney Mines Area, Nova Scotia	Harbour	Channel	H.V.A. Bit.
32993	Sydney Mines Area, Nova Scotia	Harbour, Rider	Channel	H.V.A. Bit.
B13706	Joggins Area, Nova Scotia	Forty Brine	Lump	H.V.A. Bit.
B13710	Joggins Area, Nova Scotia	Forty Brine	Pea	H.V.A. Bit.
32690	Joggins Area, Nova Scotia	Forty Brine	Slack	H.V.A. Bit.
32681	Pictou County, Nova Scotia		Lump	H.V.A. Bit.
32450	Crowsnest Area, Alberta	No. 2	Lump	M.V. Bit.
32542	Crowsnest Area, Alberta	No. 2	Stoker	M.V. Bit.
32683	Crowsnest Area, Alberta	No. 1	Lump	M.V. Bit.
32667	Crowsnest Area, Alberta	No. 2	Middlings	M.V. Bit.
32692	Crowsnest Area, Alberta	No. 2	Fines	M.V. Bit.
32707	Crowsnest Area, Alberta	No. 2	Fines	M.V. Bit.
32803	Crowsnest Area, Alberta	No. 1	Briquettes	M.V. Bit.
32735	Crowsnest Area, Alberta	No. 2	Slurry	M.V. Bit.
32737	Crowsnest Area, Alberta	No. 2	Mine Run	M.V. Bit.
32770	Crowsnest Area, Alberta	No. 2	Mine Run	M.V. Bit.
32850	Crowsnest Area, Alberta	No. 2	Mine Run	M.V. Bit.
32817	Crowsnest Area, Alberta	No. 2	Mine Run	M.V. Bit.
33011	Crowsnest Area, Alberta	Nos. 2, 4	Lump	M.V. Bit.
33036	Crowsnest Area, Alberta	Nos. 2, 4	Lump	M.V. Bit.
33056	Crowsnest Area, Alberta	Nos. 2, 4	Slack	M.V. Bit.
33101	Crowsnest Area, Alberta	Nos. 2, 4	Stoker	M.V. Bit.
33123	Crowsnest Area, Alberta	Nos. 2, 4	Fines	M.V. Bit.
33145	Crowsnest Area, Alberta	Nos. 2, 4	Slurry	M.V. Bit.
32358	Crowsnest Area, B. C.	Nos. 4, 9, 10	Slack	M.V. Bit.
32650	Mountain Park Area, Alberta		Briquettes	M.V. Bit.

[a] H.V.A. Bit. = high volatile A bituminous.
M.V. Bit. = medium volatile bituminous.

The seven different types of crucibles employed in this study were as follows: Type A—Quartz, low form, rim diam 36 mm, height 26 mm, capacity 19 ml. This crucible was used in the standard ASTM free-swelling test. Type B—Quartz, high form, rim diam 32 mm, height 38 mm, capacity 21+ ml. Type C—Platinum, 25 cc, rim diam 35 mm, height 35 mm, capacity 25 ml. Type D—Porcelain, wide form (Coors 0), rim diam 41 mm, height 25 mm, capacity 18 ml. Type E—Porcelain, high form (Coors 0), rim diam 35 mm, height 27 mm, capacity 15 ml. Type F—Nickel, high form, rim diam 36 mm, height 34 mm, capacity 28 ml. Type G—Platinum, 15 cc, rim diam 32 mm, height 32 mm, capacity 18.5 ml. This crucible conforms to the specifications of the standard ASTM volatile matter test described under ASTM Designation: D271-48, which specifies platinum crucibles of not less than 10 or more than 20-ml capacity, of not less than 25 or more than 35 mm in diam, and not less than 30 nor more than 35 mm in height.

Table II. Description of United States Coals

Sample No.	Origin	Seam	Size	Rank[a]
A501	Hopkins Co., Ky.	Nos. 9, 11	1¼″ Stoker	H.V.B. Bit.
A491	Armstrong Co., Pa.	L. Kittaning	¼ x 1¼ in.	H.V.A. Bit.
A513	Harrison Co., W. Va.	Pittsburgh	¾ x 2 in.	H.V.A. Bit.
A468	Ohio Co., W. Va.	Pittsburgh No. 8	⅝ x 1¼ in. Pea	H.V.A. Bit.
A512	Mingo Co., W. Va.	Thacker	Stoker	H.V.A. Bit.
A473	Pennsylvania		1½ x 2 in.	H.V.A. Bit.
A471	Armstrong Co., Pa.	L. Kittaning	⅜ x 1¼ in.	H.V.A. Bit.
A508	Logan Co., W. Va.	Island Creek	¼ x 1¼ in.	H.V.A. Bit.
A475	Fairmont Dist., Va.		Nut Slack	H.V.A. Bit.
A470	Floyd Co., Dist. No. 8, Ky.	Elkhorn No. 3	⅛ x1 in.	H.V.A. Bit.
A493	Letcher Co., Ky.	Elkhorn No. 3	⅛ x ⅜ in.	H.V.A. Bit.
A505	Fayette Co., W. Va.	Sewell	0 x 2 in.	H.V.A. Bit.
A525	Harlan Co., Ky.	Darby No. 5	Stoker	H.V.A. Bit.
A482	Armstrong Co., Pa.	Upper Freeport	¼ x ¾ in.	H.V.A. Bit.
32684	Armstrong Co., Pa.	Upper Freeport	Slack	H.V.A. Bit.
32886	Armstrong Co., Pa.	Upper Freeport	Slack	H.V.A. Bit.
32889	Tennessee	Upper Banner		H.V.A. Bit.
32890		Creek		H.V.A. Bit.
32891		Pond Creek		H.V.A. Bit.
32892	Buchanan Co., Va.	Clintwood		H.V.A. Bit.
32893	West Virginia	Pond Creek		H.V.A. Bit.
32894		Pittsburgh		H.V.A. Bit.
33094	Kentucky	Harlan	Stoker	H.V.A. Bit.
32885				H.V.A. Bit.
32624				H.V.A. Bit.
A495	Buchanan Co., Va.	Cary	¼ x 1¼ in.	M.V. Bit.
A484	Cambria Co., Pa., Dist. No. 1	Miller or B	¼ x ¾ in.	L.V. Bit.
A519	McDowell Co., W. Va.	Pocahontas No. 4	Stoker	L.V. Bit.
A498	McDowell Co., W. Va.	Pocahontas No. 4	⅛ x ⅝ in.	L.V. Bit.
A489	McDowell Co., W. Va.	Pocahontas No. 4	Stoker	L.V. Bit.
32888		Miller or B	Slack	L.V. Bit.
32887	West Virginia	Pocahontas No. 3	Slack	L.V. Bit.
33095		Pocahontas No. 3	Briquettes	L.V. Bit.

[a] H.V.A. Bit., high volatile A bituminous.
M.V. Bit., medium volatile bituminous.
L.V. Bit., low volatile bituminous.
H.V.B. Bit., high volatile bituminous.

Table III. Analysis of Canadian Coals

Sample No.	Total Moisture, Pct	Analysis, Dry Basis: Ash, Pct	Analysis, Dry Basis: Volatile Matter, Pct	Analysis, Dry Basis: Fixed Carbon, Pct	Rank[a] ASTM
B13612	2.1	4.3	38.3	57.4	H.V.A. Bit.
B13666	2.3	5.7	37.9	56.4	H.V.A. Bit.
B13735	1.2	5.7	35.7	58.8	H.V.A. Bit.
32625	10.8	11.8	33.0	55.2	H.V.A. Bit.
32649	2.8	10.0	33.8	56.2	H.V.A. Bit.
B13702	3.0	11.1	38.2	50.7	H.V.A. Bit.
B13704	3.6	12.3	37.7	50.0	H.V.A. Bit.
32736	8.0	17.4	34.8	47.8	H.V.A. Bit.
32920	1.9	5.6	36.9	57.5	H.V.A. Bit.
32940	2.4	5.5	36.8	57.7	H.V.A. Bit.
32958	2.3	4.3	38.0	57.7	H.V.A. Bit.
32976	2.1	4.9	37.4	57.7	H.V.A. Bit.
32993	1.7	26.4	31.5	42.1	H.V.A. Bit.
B13706	2.6	18.6	36.9	44.5	H.V.A. Bit.
B13710	2.0	31.2	30.9	37.9	H.V.A. Bit.
32690	3.5	33.9	30.0	36.1	H.V.A. Bit.
32681	2.4	13.1	29.1	57.8	H.V.A. Bit.
32450	1.2	20.5	27.8	51.7	M.V. Bit.
32542	4.3	16.0	25.5	58.5	M.V. Bit.
32683	2.8	9.0	25.5	65.5	M.V. Bit.
32667	2.2	24.0	23.9	52.1	M.V. Bit.
32692	3.1	21.5	24.1	54.4	M.V. Bit.
32707	1.4	19.2	24.3	56.5	M.V. Bit.
32803	0.5	19.1	24.2	56.7	M.V. Bit.
32735	15.2	16.9	24.2	58.9	M.V. Bit.
32737	2.4	21.4	24.8	53.8	M.V. Bit.
32770	1.2	18.8	27.5	53.7	M.V. Bit.
32850	1.3	20.5	24.6	54.9	M.V. Bit.
32817	2.2	21.9	24.8	53.3	M.V. Bit.
33011	0.8	15.8	25.1	59.1	M.V. Bit.
33036	1.4	18.0	25.6	56.4	M.V. Bit.
33056	2.6	18.0	24.7	57.3	M.V. Bit.
33101	1.1	13.3	26.0	60.7	M.V. Bit.
33123	2.3	13.3	25.2	61.5	M.V. Bit.
33145	11.3	15.5	24.1	60.4	M.V. Bit.
32358	0.5	12.6	25.4	62.0	M.V. Bit.
32650	0.7	17.2	24.7	58.1	M.V. Bit.

[a] H.V.A. Bit., heavy volatile A bituminous.
M.V. Bit., medium volatile bituminous.

These crucibles are shown in Fig. 4. The platinum and nickel crucibles have tightly fitting insert lids, whereas the quartz and porcelain crucibles have cover lids which did not fit as tightly.

For this experimental work coals with a range in free-swelling properties from 1 to 9 were used. With a view to testing as wide a range in classes of coals as possible, coals from different fields in the United States as well as in Canada were used. The coals varied in rank from high volatile B bituminous to low volatile bituminous and were both of Carboniferous and Tertiary age.

Descriptions of the Canadian coals tested, showing origin, seam, size and rank, are given in Table I, and Table II presents similar data for the United States coals. Tables III and IV give the proximate analyses on the dry basis for the Canadian and United States coals.

Details of Tests

Vertical Tube Furnace: As a vertical tube furnace, often referred to as the Fieldner furnace, is used for standard volatile matter determinations at 950°C, and a low temperature tube furnace is used for determining the swelling index of coals by the Canadian Fuel Research Laboratories method,[1] a similar type of furnace first was employed in experiments to determine whether an electric furnace or heater could be used to replace the gas burner specified in the ASTM standard test for determining the free-swelling index of coal.

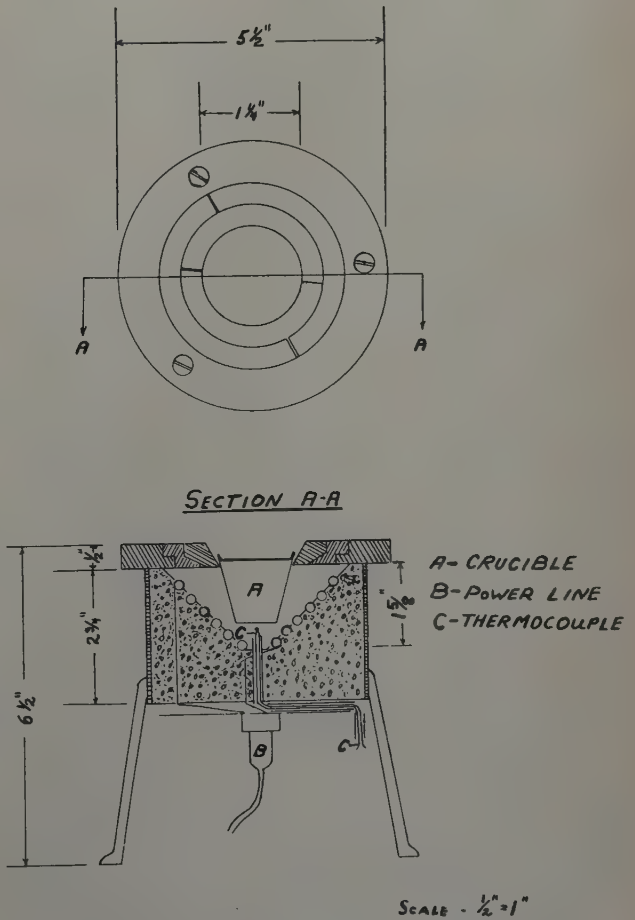

Fig. 2—Diagram of cone heater.

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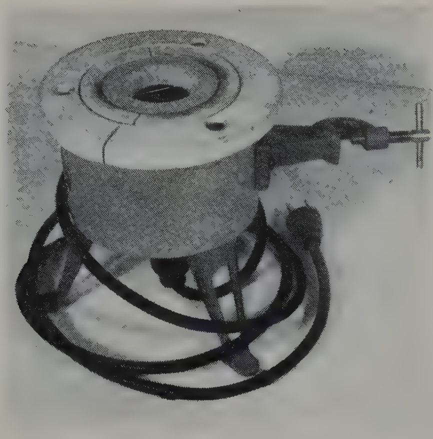

Fig. 3—Cone heater.

Two types of crucibles were used in these tests, type A, the quartz crucible recommended for the standard test, and type D, a porcelain wide form crucible somewhat similar in dimensions and capacity to the standard quartz crucible. Tests were conducted at seven different temperatures in 50°C steps from 450°C to 750°C. The temperature was that of the furnace and was taken by means of a noble metal thermocouple, the end of which was situated about ⅛ in. from the bottom of the crucible.

The free-swelling index results on 19 coals varying in index from 3 to 8 by the standard method are shown in Tables V and VI, Table V giving the results at different temperatures using the standard quartz crucible, and Table VI the results using the wide form porcelain crucible.

Tests with Cone Heater: In view of the unsatisfactory results obtained with the vertical tube furnace and because the electric cone heater is considered to be a substitute for the gas burner, this latter type was used in all subsequent experimental work.

Table IV. Analysis of United States Coal

Sample No.	Total Moisture, Pct	Analysis, Dry Basis Ash, Pct	Analysis, Dry Basis Volatile Matter, Pct	Analysis, Dry Basis Fixed Carbon, Pct	Rank[a] ASTM
A501	9.1	7.2	42.0	50.8	H.V. Bit.
A491	3.6	9.1	38.6	52.4	H.V.A. Bit.
A513	2.2	9.1	38.5	51.6	H.V.A. Bit.
A468	3.4	11.1	37.7	51.2	H.V.A. Bit.
A512	2.5	5.9	37.5	56.6	H.V.A. Bit.
A473	1.7	9.4	36.7	53.9	H.V.A. Bit.
A471	2.6	9.2	36.5	54.3	H.V.A. Bit.
A508	4.4	7.4	36.6	56.0	H.V.A. Bit.
A475	4.5	9.5	35.6	54.9	H.V.A. Bit.
A470	2.5	6.8	35.3	57.9	H.V.A. Bit.
A493	5.7	5.2	34.7	60.1	H.V.A. Bit.
A505	6.0	9.8	34.3	55.8	H.V.A. Bit.
A525		3.9	34.2	61.9	H.V.A. Bit.
A482	3.4	10.9	32.1	57.0	H.V.A. Bit.
32684	1.3	11.3	33.2	55.5	H.V.A. Bit.
32886	3.7	11.5	34.9	53.6	H.V.A. Bit.
32889	1.1	8.4	32.2	59.4	H.V.A. Bit.
32890	2.0	7.9	36.1	56.0	H.V.A. Bit.
32891	2.3	6.3	38.7	55.0	H.V.A. Bit.
32892	1.8	7.2	33.3	59.5	H.V.A. Bit.
32893	2.1	6.8	35.9	57.3	H.V.A. Bit.
32894	1.4	8.7	34.0	57.3	H.V.A. Bit.
33094	4.7	4.4	39.5	56.1	H.V.A. Bit.
32885	16.8	8.7	37.0	54.3	H.V.A. Bit.
32624	7.0	9.1	34.3	56.6	H.V.A. Bit.
A495	3.1	10.1	21.3	68.6	M.V. Bit.
A484	1.9	8.8	17.3	73.9	L.V. Bit.
A519	1.2	8.3	15.4	76.3	L.V. Bit.
A498	2.4	6.9	15.2	77.9	L.V. Bit.
A489	2.6	6.3	15.1	78.6	L.V. Bit.
32888	0.5	8.5	18.1	73.4	L.V. Bit.
32887	0.7	9.1	17.8	73.1	L.V. Bit.
33095	0.3	7.2	24.1	68.7	L.V. Bit.

[a] H.V. Bit., heavy volatile bituminous.
H.V.A. Bit., heavy volatile A bituminous.

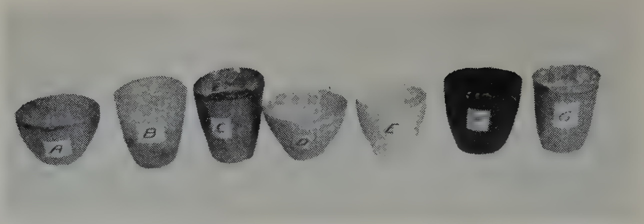

Fig. 4—Crucibles.

A—Quartz, low form, standard free-swelling test
B—Quartz, high form
C—Platinum, 25 cc
D—Porcelain, wide form
E—Porcelain, high form
F—Nickel, high form
G—Platinum, 15cc, standard volatile matter test

The first series of tests with this furnace or heater was conducted at two temperatures, 600°C and 650°C. The temperature is that taken inside at the bottom of the crucible by means of a noble metal thermocouple. Six different types of crucibles, A, B, C, D, E and F, were employed, and two different heating times were used, 4 min and 15 min.

The results of these tests on seven different coals

Table V. Tests in Vertical Tube Furnace, Free-Swelling Index at Varying Temperatures Using the Quartz Crucible Type A,[a] See Fig. 5

Furnace Temp., °C Sample No.	450 S.I.[b]	500 S.I.[a]	550 S.I.[a]	600 S.I.[a]	650 S.I.[a]	700 S.I.[a]	750 S.I.[a]	Standard Test S.I.[a]
A501	2	2	3	3	3	2.5	3	4+
A491	3	3.5	5.5	4.5	4	3.5	3	8
A513	3	4	5	4.5	5	4	3.5	7.5
A468	2	3.5	4.5	4.5	3.5	2.5	2.5	6
A512	3.5	3	4	4	3.5	2.5	2.5	5+
A473		9	8	5	4	3.5	3	8
A471	4	4	6.5	4.5	4	4.5	4	7.5
A508	1.5	2	3.5	5	4.5	3.5	3	6.5
A475	1.5	2.5	5	6	4.5	4	3	8
A470	0	2	3	3	2.5	2.5	2	3
A493	1	1.5	1.5	2.5	2	2	2	3
A505	1	1	1	1.5	2	1.5	1.5	3
A525	2	2	2.5	2.5	2	2	1.5	2.5
A482	9	9	9	5	4	4		8.5
A495	0	0	4	5.5	4	3	3.5	4.5
A484	0	1	1	1.5	3	3	3	4.5
A519	0	2	2	2.5	3.5	2.5	3	4.5
A498	0	0	1	1	2	2	3	4.5
A489	0	0	0	3.5	4	4.5	4	5.5

[a] Same crucible as used in standard test.

[b] Swelling index as determined by comparison of buttons to standard profiles.

Table VI. Tests in Vertical Tube Furnace, Free-Swelling Index at Varying Temperatures Using Porcelain, Wide Form, Crucible Type D, See Fig. 6

Furnace Temp., °C Sample No.	450 S.I.[a]	500 S.I.[a]	550 S.I[a]	600 S.I.[a]	650 S.I.[a]	700 S.I.[a]	750 S.I.[a]	Standard Test S.I.[a]
A501	1	1	1.5	2	2	2	2	4+
A491	2	3.5	5	6.5	6	5	4	8
A513	1.5	3	4.5	5.5	5	4.5	4	7.5
A468	1.5	2.5	3.5	4.5	4	3.5	3	6
A512	1	1.5	2	4	3.5	3.5	2	5+
A473	3	4	5.5	8.5	7	4.5	5	8
A471	1	1.5	3.5	5	5	5	4	7.5
A508	1	2	3	4	4.5	4	2.5	6.5
A475	1	2.5	3.5	5	6	5.5	4	8
A470	0	1	1	1.5	2	1.5	2	3
A493	0	1	1.5	2	2	2	2	3
A505	1	1	1.5	1.5	2	2	2	3
A525	1	1	1	1.5	2	1.5	1.5	2.5
A482	2.5	3	5	9	8.5	7.5	4	8.5
A495	0	2	2	3	4	6	5	4.5
A484	0	1	0	1	1.5	2	3	4.5
A519	0	0	1	1	2	3	3	4.5
A498	0	0	0	1	1.5	2	3	4.5
A489	0	0	0	1.5	3	4	4	5.5

[a] Swelling index as determined by comparison of buttons to standard profiles.

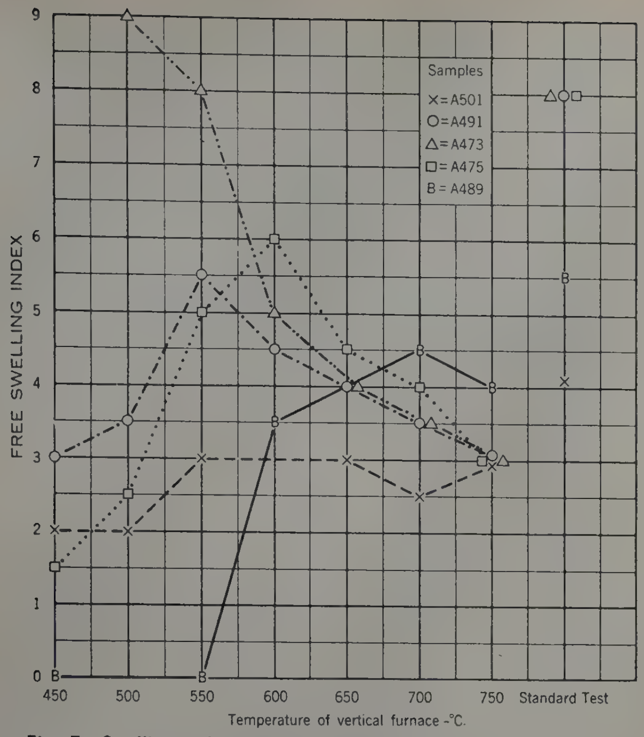

Fig. 5—Swelling indices at various temperatures using quartz crucible A, see Table V.

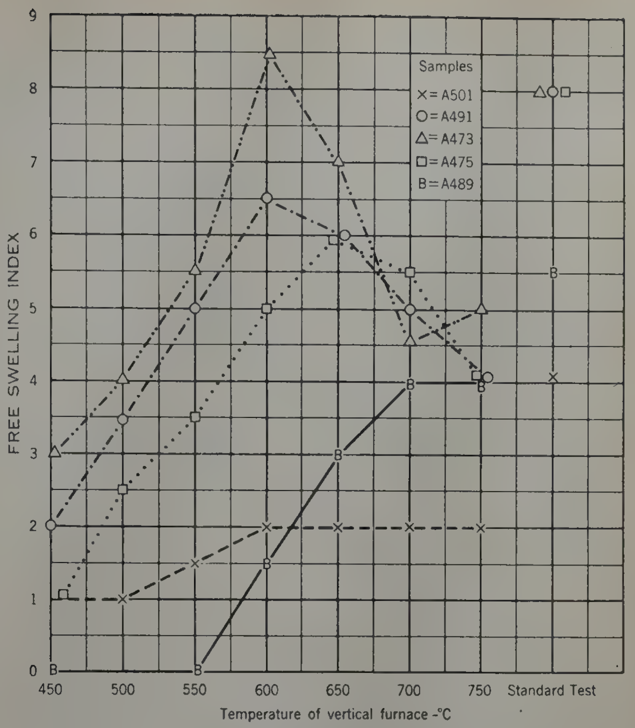

Fig. 6—Swelling indices at various temperatures using porcelain crucible D, see Table VI.

varying in free-swelling index from 3 to 8 are shown in Table VII.

The second series of tests with the cone heater was conducted at 600°C, 650°C, 700°C, and 820°C, using only the type A quartz crucible which is the type used in the standard test. The results of these tests on 17 coals are shown in Table VIII.

The third series of tests was conducted at two temperatures, 600°C and 650°C using the type G, 15-cc platinum crucible, which is the type recommended for use in the standard volatile matter test. The heating time in these tests was 4 min. The results on 24 coals, varying in free-swelling index from 2.5 to 8.5, are presented in Table IX.

The results of the above tests established the type of equipment and method to be employed in conducting a test using electrical heaters to give results comparable to those obtained with the standard gas burner method. To check how closely the results determined by the electric method compare with the results as determined by the standard method, a comparative series of determinations was conducted using the 350-w cone heater at 650°C and type G, 15-cc platinum crucibles with a 4-min heating period. The results obtained on 30 Canadian coals are shown in Table X, and similar results on 30 United States coals are presented in Table XI.

Discussion of Results

Vertical Tube Furnace Tests: The first series of tests using the electric vertical tube furnace was conducted to determine at what temperature, if any, the determination might be conducted using the standard quartz crucible, type A, to obtain buttons

Table VII. Tests with 350-W Cone Heater
Free-Swelling Index in Various Types of Crucibles at 600°C and 650°C With Varying Duration of Heats, See Figs. 7 and 8

Sample No.	Temp. in Crucible, 600°C						Temp. in Crucible, 650°C						Standard Test
	Type of Crucible[a]						Type of Crucible[a]						
	A	B	C	D	E	F	A	B	C	D	E	F	
A501													
4 min	1.5	2.5	2.5	2	2.5	1	2	3	2.5	2	3	2	4+
15 min	1.5	3	1.5	2	2.5	1	1.5	2.5	2.5	2.5	3	1.5	
A468													
4 min	3.5	6.5	5	5	5	3.5	4	6.5	3.5	4	5.5	4.5	6
15 min	3.5	6.5	5	5	4.5	3.5	4	6.5	7	5	5.5	2.5	
A512													
4 min	3	4	3	3.5	4	2	3	4.5	3	3.5	5	2.5	5+
15 min	2.5	3.5	2	3	4	1.5	2.5	5	3	3.5	4	2	
A473													
4 min	5.5	7	8	7	8	4.5	5.5	8	8	7.5	8.5	9	8
15 min	5	7	8	5.5	8	8.5	5.5	8	8	8.0	8	7.5	
A508													
4 min	3.5	5.5	4	4	5	2.5	4	6	4.5	4.5	5	2	6.5
15 min	3	5	3	4	4.5	2	3.5	6	6.5	4	5	2.5	
A470													
4 min	1.5	2.5	2	2	2	1	1.5	3	2	2	2	1	3
15 min	1.5	2	2	2	1.5	1	2	3	3	2.5	2	1	
A482													
4 min	4.5	7	8.5	7	7	7.5	6	8	8.5	7.5	7.5	8	8.5
15 min	5	6.5	9	5.5	7.5	5	5	8	8.5	7.5	7.5	9	

[a] Types of Crucibles: A—quartz, low; B—quartz, high; C—platinum, 25cc; D—porcelain, wide; E—porcelain, high; F—nickel, 28cc.

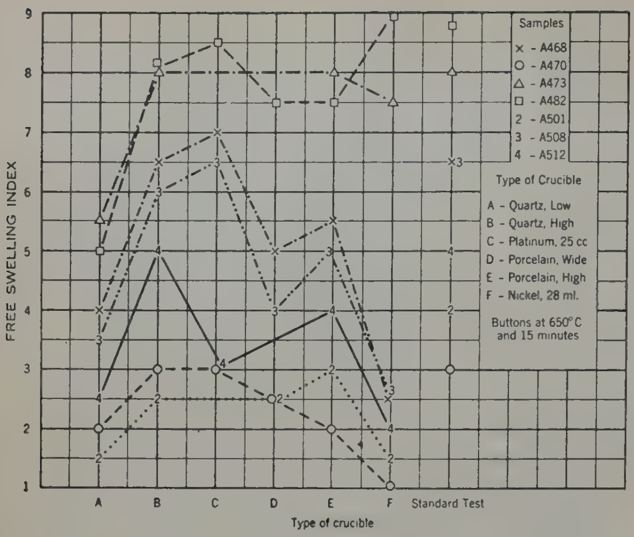

Fig. 7—Swelling index in various types of crucibles at 650°C in cone heater, see Table VII.

giving the same free-swelling index as when conducted by the standard gas burner method. The results shown in Table V are presented in graphic form in Fig. 5.

No two coals appeared to react the same. Some showed their highest swelling at the lower temperatures and some at the highest temperatures employed. Only in two cases out of 19 were values obtained which checked the free-swelling indices obtained by the standard method. The results are fairly conclusive in indicating that this combination of equipment was unsatisfactory for the purpose.

The results of the second series of tests using the same furnace and the type D, wide form, porcelain crucible are shown in Table VI, and presented in graphic form in Fig. 6. Again only two coals out of the 19 tested showed free-swelling values comparable to those obtained by the standard test. The results of this series also conclusively indicate that the above combination of equipment was unsatisfactory as a substitute for the gas burner method.

Cone Heater Tests: To determine whether and under what conditions the electric cone heater, considered to be a substitute for the Bunsen burner, could be employed to replace the standard gas burner method three series of tests were conducted.

Table VIII. Tests with 350-W Cone Heater Free-Swelling Index in Quartz Crucible Type A at Varying Temperatures, See Figs. 9 and 10

Sample No.	600°C S.I.[b]	650°C S.I.[b]	700°C S.I.[b]	820°C S.I.[b]	Standard Test S.I.[b]
A501	1.5	2	2.5	4	4+
A491	—	—	7.5	8	8
A513	—	—	7	7.5	7.5
A468	3.5	4	5.5	6.5	6
A512	3	3	4	5.5	5+
A473	5	5.5	8	8	8
A471	—	—	5.5	7	7.5
A508	3.5	4	5.5	6	6.5
A475	—	—	5.5	7.5	8
A470	1.5	2	2	3	3
A493	—	—	2.5	3	3
A505	—	—	2	3	3
A525	—	—	2	3	2.5
A482	0.5	5.5	8	8.5	8.5
A495	—	—	4	4.5	4.5
A484	—	—	2.5	4	4.5
A519	—	—	3	4	4.5

[a] Final temperature to be attained in standard test.
[b] Swelling index as determined by comparison of buttons to standard profiles.

Table IX. Test with 350-W Cone Heater. Comparison of Free-Swelling Index Results at 600°C and 650°C (Conducted in Type G, 15-cc Platinum Crucibles with 4-Min Heating) to Those by Standard Test, See Figs. 11 and 12

Sample No.	Using 15-cc Platinum Crucible in Cone Heater Free Swelling Index 600°C	650°C	Standard ASTM Test Free Swelling Index
A501	3.0	3.5	4.0
A491	7.5	8.0	8.0
A513	7.0	7.5	7.5
A468	6.5	6.0	6.0
A512	5.0	4.5	5.0
A473	8.5	8.0	8.0
A471	7.0	7.5	7.5
A508	6.5	6.5	6.5
A475	7.5	7.5	8.0
A470	2.5	3.0	3.0
A493	2.5	2.5	3.0
A505	2.0	2.0	3.0
A525	2.0	2.0	2.5
A482	9.0	9.0	8.5
A495	7.5	7.5	8.0
A484	3.5	4.5	4.5
A519	4.0	4.5	4.5
B13612	7.5	7.0	7.5
B13666	6.5	6.5	7.0
B13735	7.5+	7.5+	8.5
B13702	5.0	5.5+	6.0
B13704	5.0	5.5+	6.0
B13706	4.0	5.0	5.5
B13710	1.5	2.0	2.5

The first series was conducted at 600°C and 650°C using six different types of crucibles, the standard quartz, type A; a quartz high form, type B; a 25-cc platinum, type C; a wide form porcelain, type D; a high form porcelain, type E; and a high form nickel crucible, type F. Two subseries of tests were run using different heating periods, one at 4 min and the other at 15 min. The results of all these tests on seven different coals are shown in Table VII, with the results obtained at 650°C and 15 min heating being presented graphically in Fig. 7. A photographic record of the buttons for three of the coals is shown in Fig. 8.

First, the data indicate that a temperature of

Table X. Comparison of Free-Swelling Index as Determined with 350-W Cone Heater at 650°C Using 15-cc Platinum Crucibles and 4-Min Heating Period to Results of Standard ASTM Test Canadian Coals

Sample No.	Free-Swelling Index Standard Test Index	Volume of Button, ml	Cone Heater Test Index	Volume of Button, ml
32625	7.0	4.7	7.0	4.9
32649	8.0	5.9	7.8	5.2
32736	3.4	2.1	3.1	2.7
32920	6.5	4.9	6.5	4.9
32940	6.1	4.5	5.8	4.4
32958	6.5	4.8	6.4	5.1
32976	6.0	4.7	6.1	4.7
32993	5.5	3.1	5.6	3.9
32690	1.0	1.3	1.0	1.8
32681	4.8	3.1	4.0	3.1
32450	3.5	2.1	3.5	2.5
32542	2.0	1.6	1.5	2.1
32683	5.9	4.6	5.8	5.2
32667	1.5	1.1	1.5	1.4
32692	1.5	0.8	1.5	1.5
32707	1.5	1.3	1.5	1.9
32803	1.0	1.3	1.0	1.5
32735	1.0	1.1	1.0	1.4
32737	2.0	1.8	1.5	1.9
32770	3.5	2.5	2.8	2.8
32850	2.5	1.8	2.5	2.3
32817	1.6	1.5	1.5	1.8
33011	3.5	2.4	3.0	2.2
33036	3.5	2.3	4.1	3.2
33056	4.0	2.5	4.0	3.3
33101	3.5	2.5	3.5	3.5
33123	4.0	3.1	3.9	3.7
33145	3.0	2.2	3.2	2.6
32358	8.0	6.4	8.0	7.2
32650	1.0	1.3	1.5	1.4

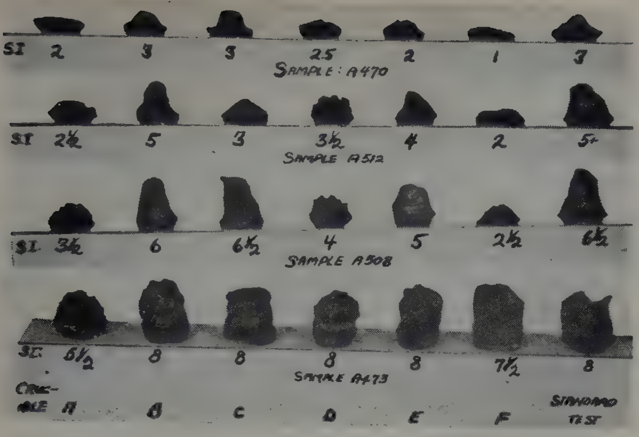

Fig. 8—Comparison of coke buttons prepared at 650°C in various types of crucibles using a cone heater, see Table VII.

650°C appears to give results approaching a little more closely those obtained by the standard method than does a temperature of 600°C. In addition, heating for a period longer than 4 min does not appear to make any appreciable difference in most cases. Of all the crucibles tested the type B high form quartz and the 25-cc platinum type C gave the most consistent results in comparison to the standard, with the platinum crucible showing slightly more consistency.

To determine whether the standard quartz crucible type A might be suitable at some different temperature than those used above, a second series of tests was run on 17 coals at temperatures of 600°C, 650°C, 700°C, and 820°C, the latter being the temperature specified in the standard gas burner method. The results of these tests are shown in Table VIII and are presented graphically in Fig. 9. A photographic record of the buttons from four of the coals tested is shown in Fig. 10. These results show conclusively that the standard quartz crucible when used in the cone heater at 820°C, the standard heating temperature, gives results close to those obtained by the standard gas burner method. Thus all other conditions remaining the same, the cone heater may be used to replace the gas burner.

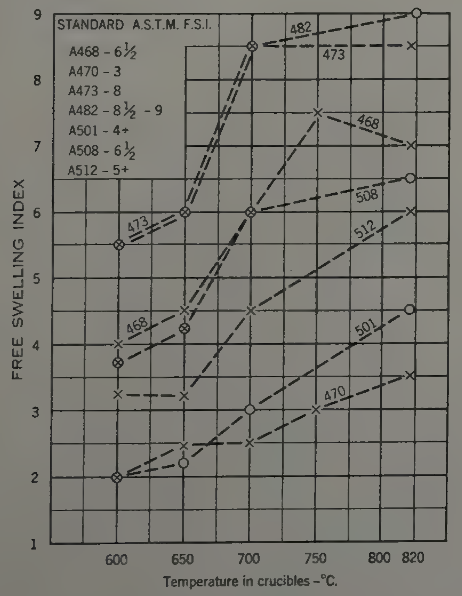

Fig. 9—Free-swelling index in quartz, type A crucible at varying temperatures, see Table VIII.

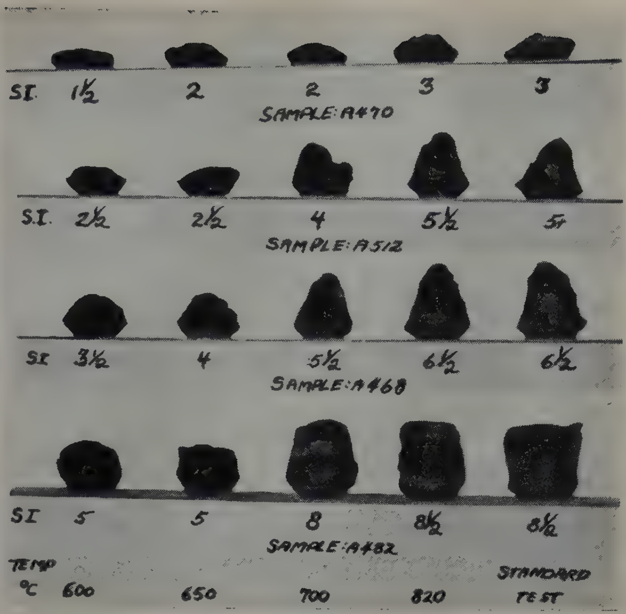

Fig. 10—Comparison of coke buttons prepared in crucible type A (quartz) at varying temperatures, see Table VIII.

However, as the quartz crucible is not usually available as standard equipment, a third series of tests was conducted to determine whether the 15-cc platinum crucible, type G, used in determining volatile matter by the standard ASTM method, would be suitable to replace the quartz crucible. Preliminary experimentation indicated that a temperature of 600°C to 650°C would be suitable for platinum crucibles in comparison to 820°C for quartz because of the comparatively high heat conductivity of platinum. A series of tests was conducted at these two temperatures with a 4-min heating period. The results obtained on 24 Canadian and United States coals of varying rank and with free-swelling indices

Table XI. Comparison of Free-Swelling Index as Determined with 350-W Cone Heater at 650°C Using 15-cc Platinum Crucibles and 4-Min Heating Period to Results of Standard ASTM Test, United States Coals

	Free-Swelling Index			
	Standard Test		Cone Heater Test	
Sample No.	Index	Volume of Button, ml	Index	Volume of Button, ml
A501	4.0		3.5	
A491	8.0		8.0	
A513	7.5		7.5	
A468	6.0		6.0	
A512	5.0		4.5	
A473	8.0		8.0	
A471	7.5		7.5	
A508	6.5		6.5	
A475	8.0		7.5	
A470	3.0		3.0	
A493	3.0		2.5	
A505	3.0		2.0	
A525	2.5		2.0	
A482	8.5		9.0	
32684	8.1	6.9	8.4	7.3
32886	8.0	7.2	7.5	7.2
32889	7.0	5.2	7.0	5.2
32890	5.4	4.1	5.1	3.7
32891	5.5	4.0	5.0	3.9
32892	7.5	6.5	7.5	5.8
32893	6.5	4.7	5.4	4.7
32894	7.6	5.7	7.5	6.1
33094	4.3	2.7	4.4	3.5
32885	5.0	4.2	5.5	4.8
A495	8.0		7.5	
A484	4.5		4.5	
A519	4.5		4.5	
32888	7.0	5.0	6.1	4.7
32887	7.5	6.9	7.6	6.4
33095	9.0	8.6	8.5	8.6

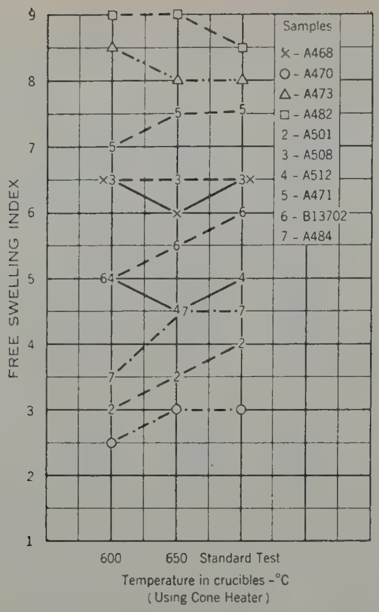

Fig. 11—Swelling indices at varying temperatures in 15-cc platinum crucibles in comparison to standard test results.

varying from 2.5 to 8.5 are shown in Table IX, the results on some of the coals being shown graphically in Fig. 11. In addition, a photographic record of the buttons from four of the coals tested is shown in Fig. 12.

These results indicate conclusively that at a temperature of 650°C, inside the crucible, free-swelling index values are obtained which check very closely the results obtained by the standard gas burner method irrespective of the rank of the coal, its origin, or its swelling properties.

To determine whether consistently comparable results could be obtained over a period of time when the method (cone heater, 650°C, 15-cc platinum crucible, 4-min heating time) is used in the regular routine of the laboratory, the free-swelling indices of 30 Canadian and 30 United States coals were determined in parallel with the standard gas burner method. The results shown in Tables X and XI, indicate that the cone heater method gives free-swelling values comparable to those obtained by the gas burner method, irrespective of the free-swelling properties of the coals, their rank, origin, or geological age.

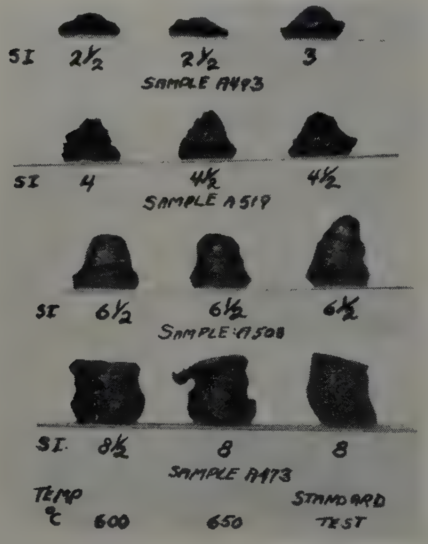

Fig. 12—Comparison of coke buttons at 600°C and 650°C prepared in 15-cc platinum crucible, type G, using a cone heater, see Table VII.

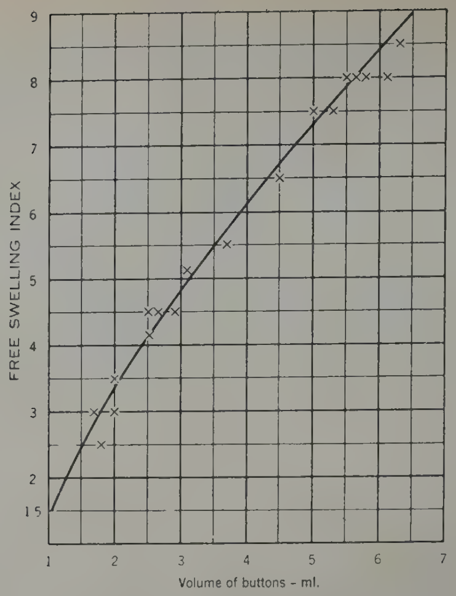

Fig. 13—Relationship of volume of buttons to free-swelling index, standard test.

As the free-swelling indices of the coals are obtained by comparing the buttons to a series of standard profiles, a further check on the comparison between the size of the buttons as obtained by the cone heater and gas burner methods was desirable. Thus the volumes of the buttons were determined. This was done in the following manner. First the volume of the crucibles was measured with 150x200 mesh sand pouring the sand into the crucibles from a funnel with the outlet end placed exactly 3¾ in. above the rim of the crucible. The crucible full of sand was tapped six times, scraped level and then poured into a 10-cc graduated cylinder, tapped six times and the volume obtained. After running a test and before removing the button from the crucible, sand is poured into the crucible in the manner described above until the void is filled. The sand volume is measured as above. The volume of the button then is:

Volume of crucible — volume of void.

Table XII presents data on a series of coals showing the relationship of the free-swelling index to the volume of the buttons as obtained by using the standard ASTM gas burner method. Fig. 13 presents the relationship in a graphic manner, the volume of the buttons being plotted against the free-swelling indices. The almost straight line obtained indicates that the index is directly proportional to the volume of the buttons.

In Tables X and XI, in addition to the free-swelling indices, the average volumes of the buttons, four for each determination, are shown both as obtained by the standard test and by the modified cone heater test. It is of importance to note that the volume of the buttons as obtained by the two methods check very closely, as do the free-swelling indices. Thus it may be concluded that for all practical purposes the index as obtained by comparison to the standard profiles is sufficiently accurate to define the free-swelling properties of the coal.

Occurrence And Exploration Of Georgia's Kaolin Deposits

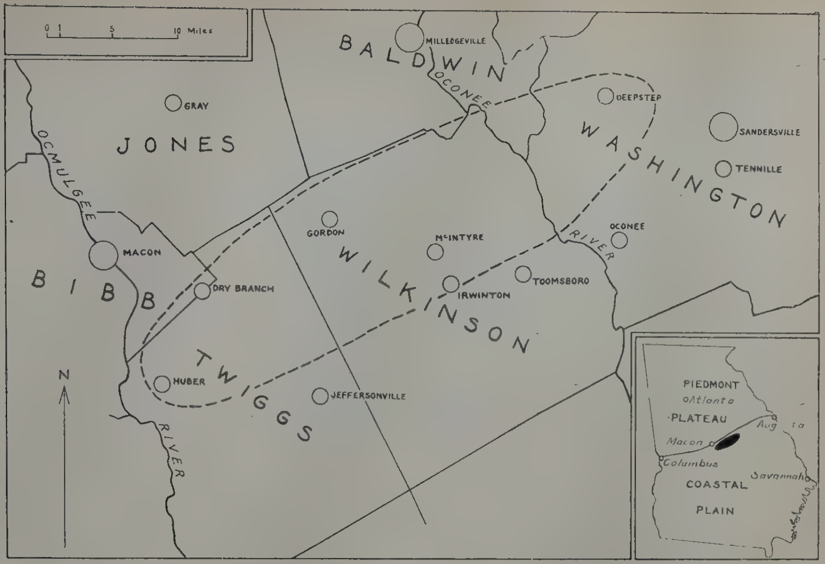

Fig. 1—Principal kaolin-producing area of Georgia, with index map showing relation of the area (black) to the Coastal Plain and the Piedmont Plateau.

by Thomas L. Kesler

If all of the 14 million tons of kaolin produced in Georgia through 1949 had been mined from a single deposit 20 ft thick, it would represent a mined-out area of less than 1 sq mile. This measure of depletion is in such strong contrast with the large size of the area shown in Fig. 1, from which most of the output has come, that there has developed an optimistic outlook concerning future supplies of kaolin equal in grade to that already mined. Dozens of mines are widely but not evenly distributed within this area, which consists mostly of parts of Twiggs, Wilkinson, and Washington counties. There are scarcely half a dozen kaolin mines elsewhere in the state that have been appreciably productive, although the areas in which they occur may have good possibilities. Most of the efforts to find new deposits, therefore, will be confined to the area shown in Fig. 1, for it is here that the industry has established its facilities, and prospecting is encouraged by relatively numerous outcrops and opened deposits of kaolin.

The Ocmulgee and Oconee rivers drain the area. They and their main tributaries have flat, swampy floodplains covered with alluvium and for the most part thickly wooded. The divides between these streams are intricately dissected by the minor tributaries, which are spring-fed in steep-walled, cirque-like heads. Gullying and under-cutting are active along the streams but are partly controlled by unusually dense vegetation. The mean altitude is about 350 ft above sea level, and locally the relief ranges up to 200 ft.

The area is near the inner margin of the Coastal Plain, and is underlain by Cretaceous and Tertiary formations. Its general location is shown in black in Fig. 1. The division between the Coastal Plain and the Piedmont plateau is actually a very sinuous line, the fall line, marking the extent to which erosion has stripped the Coastal Plain sediments from the southeastward-sloping surface of the crystalline rocks of the Piedmont.

Cretaceous Host Formation

The kaolin deposits, which are described in the succeeding section, occur sporadically in lenslike bodies in a thick series of poorly sorted clayey sands constituting the lowermost of the Upper Cretaceous formations, the Tuscaloosa. The basis for this correlation has been reviewed by Cooke.[1] The same formation has been termed the Potomac group by Ladd[2] and the Middendorf formation by Smith[3] in reports on the kaolin deposits. In another report, Veatch[4] called the formation the Tuscaloosa but correlated it with the Lower Cretaceous.

The Tuscaloosa in this area consists of fine to very coarse sands, commonly cross-bedded, which contain widely different proportions of white, pink, and yellow kaolin and considerable white mica. A few of the coarser beds contain quartz grains up to ¾ in. in length, with bodies of pure white kaolin of equal size that appear to have been formed by the weathering of feldspar grains in place. In the kaolin-producing area, gravel and cobbles occur mainly at the base of the formation. Fig. 2 shows the relations of the Tuscaloosa and its kaolin deposits to the adjacent formations. Its base, on the eroded surface of the crystalline rocks, dips southeastward 50 to 60 ft to the mile, according to LaMoreaux.[5] The upper surface of the Tuscaloosa is also an unconformity,

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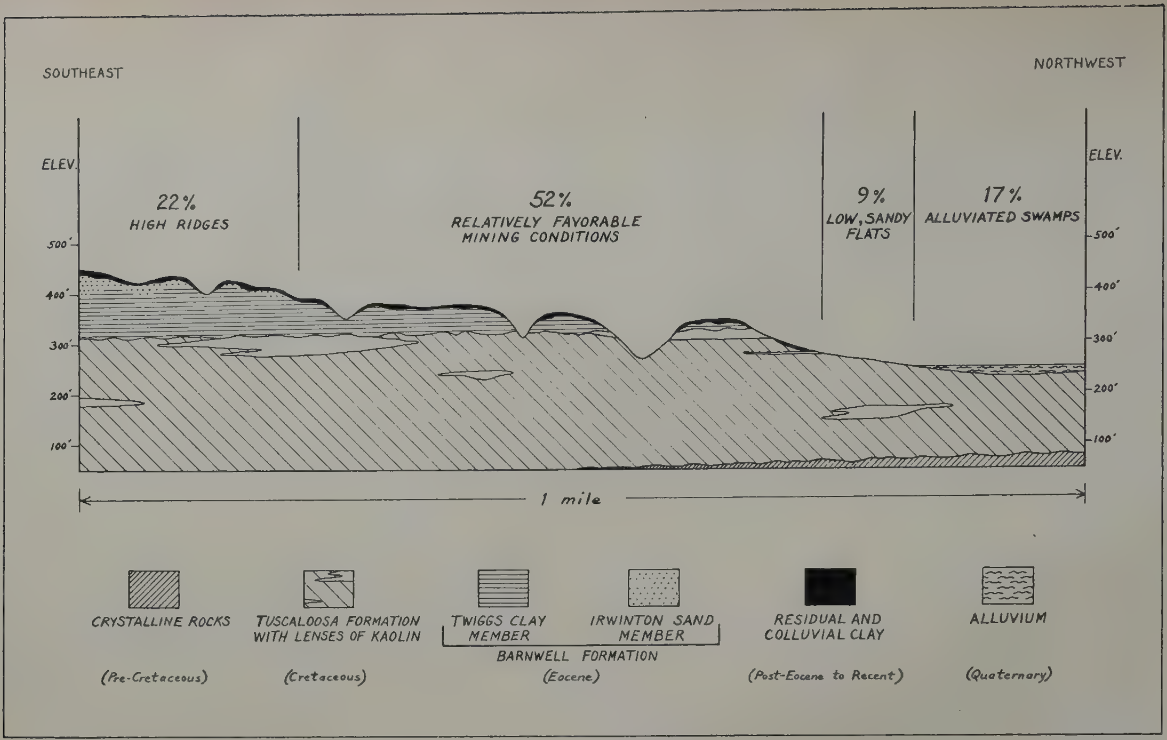

Fig. 2—Cross-section showing relations and typical occurrence of the Georgia kaolin deposits. Vertical scale is exaggerated.

and the maps of Warren and Thompson[6, 7, 8] indicate that this surface strikes about N. 45°E. and dips southeastward 11 to 15 ft to the mile. As the upper surface of the Tuscaloosa dips less steeply than the base, the formation thickens southeastward from zero along the fall line to about 600 ft[5] along the southeast edge of the area shown in Fig. 1.

Eocene Overburden Formation

The Barnwell formation, of Eocene age, rests on the Tuscaloosa, and its beds appear to have the same strike and dip as does the unconformity at its base. In the kaolin-producing area, the Tuscaloosa is exposed mostly in valley lands, where the Barnwell has been removed by erosion, as shown in Fig. 2. Owing to the intricate stream system, the contact between the formations is too complex to be shown on any map small enough to accompany this paper, and the reader is referred to the excellent maps of Warren and Thompson.[6, 7, 8]

The Barnwell consists of two members that intergrade vertically but not everywhere at the same stratigraphic level, and consequently there must also be considerable horizontal intergrading. The lower member, the Twiggs clay, consists mostly of fullers earth, in many places unevenly sandy and calcareous and containing random beds of fossiliferous limestone as much as 4 ft thick. These limestone beds, containing the fauna of the Ocala limestone, occur interbedded with the fullers earth from the base of the member upward as much as 65 ft. Hard, sandy, calcareous beds are most numerous in the lower 15 ft of the member. They yield readily to weathering along outcrops, where they have been altered to sand of different colors, which has been commonly noted in lithologic descriptions of the lower part of the Twiggs clay.

The "channel sands" above the Tuscaloosa, described by Warren and Thompson[6, 7, 8] and by LaMoreaux,[5] are probably in places these leached sandy beds at the base of the Twiggs clay, but in other places they probably include both the residual and colluvial clay, described below, and parts of the Tuscaloosa itself. No basis has been found, in the present work, for recognizing channel sands as a stratigraphic unit. All of the Twiggs clay except the purer limestone beds is olive green to greenish yellow if leached, and dark, bluish gray if relatively unweathered. The Twiggs clay member is about 80 ft thick over much of the area, but was found to be 115 ft thick in a hole drilled 5.2 miles northeast of McIntyre. The member has been studied in detail by Shearer,[9] as the fullers earth is of commercial grade in some places.

With the occurrence of thin beds of sand in its upper part, the Twiggs clay grades into the overlying member, the Irwinton sand. This member consists largely of white to yellow sand, but contains a few, thin, waxy, red to gray beds that may be much weathered fullers earth. The least weathered of the sand is loosely cemented with a white, noncalcareous material, evidently silica.

Younger Overburden Formations

A mantle of deep-red sandy clay as much as 10 ft thick overlies the two members of the Barnwell as well as parts of the Tuscaloosa, regardless of altitude, see Fig. 2. Locally, it contains transported fragments of limonitic hardpan, itself a product of weathering. In some of the mine openings the red, sandy clay can be seen forming directly from the weathering of sandy, calcareous fullers earth in the lower part of the Twiggs clay, and slumping has carried some of it down over adjacent parts of the Tuscaloosa. However, where the red, sandy clay covers the Irwinton sand high on the divides, the origin differs. In this position, the clay is undoubtedly the residuum of beds destroyed by weathering, possibly fullers earth beds that were in or that overlay the Irwinton sand. Although the red mantle is physically continuous from divides to valleys, it appears that its source is diverse. Development of the clay began after Eocene time and is still going on.

Probably the youngest of the formations associated with the kaolin deposits is the alluvium that has been deposited on the wider flood-plains, converting them into swamps. Little is known about its character or thickness, but the alluvium must be relatively thin. Most of it overlies the Tuscaloosa, occurring at altitudes below the Tuscaloosa-Barnwell contact. Its age is Quaternary, and the heavily forested condition over much of it suggests that there has been little addition of sediment during modern time.

Character of the Deposits

Lenslike deposits of white kaolin, in places mottled with yellow and red iron oxide, occur sporadically in all parts of the Tuscaloosa formation in this area. They commonly grade into the enclosing clayey sands, but some contacts are sharp. Most of the deposits are too sandy and micaceous to be of present economic interest, but the purer bodies are the basis of the kaolin industry in Georgia. Most of the purer raw kaolin from these deposits averages (dry basis) 44 to 47 pct SiO_2, 37 to 41 pct Al_2O_3, 1 to 2 pct TiO_2, less than 1 pct Fe_2O_3, and extremely small amounts of lime, magnesia, soda, and potash.[10] The freshly cored material contains 18 to 22 pct free moisture. Variations in analyses of the raw kaolin, given in the reports cited herein, are caused chiefly by differences in the proportions of fine sand and mica in the kaolin.

In the mining area, the kaolin is termed hard, semihard, or soft according to its toughness when air-dried. Kaolinite is the chief constituent, with quartz and muscovite the main impurities, and iron and titanium oxide minerals, zircon, and tourmaline the most persistent minor impurities. The deposits range from a few feet to perhaps half a mile in major dimension. A single deposit may consist of soft, semihard, or hard kaolin, or it may contain a lower layer of soft kaolin overlain by a layer of harder kaolin. Where two layers occur, there is commonly a sharp line of contact between them. Klinefelter and others[10] have shown that the particle size of the hard kaolin is finer than that of the soft kaolin.

The deposits contain a few concretions of iron sulphide ranging in size from scarcely visible pellets to several inches across, and the kaolin adjacent to the concretions is brown, apparently carbonaceous. Weathering converts the concretions into limonitic balls usually hollow or containing partly oxidized sulphide. In places, the concretions are attached in sinuous, linear groups that are mostly vertical. Those that occur at the top of a bed of kaolin, under relatively thin overburden, have been leached leaving sinuous rusty cavities partly filled with inwashed sand. These are not fossil tree roots, a possibility implied by Smith.[8]

Yellow, iron-oxide stains and black organic stains are fairly common in outcrops of the kaolin deposits and in their uppermost parts where the overburden is very thin. Such stains are clearly of post-Tertiary origin, as they diminish to extinction with increase in the thickness of cover. Drilling has shown that the iron-oxide stains, which are derived from the weathering of the iron sulphide and are the more persistent, are rarely found under 50 ft or more of overburden, and are not prominent if the cover is only 25 ft thick. They are not to be confused with the vivid brick-red iron-oxide stains, which, in many places, occur at considerable depth, and show no relation to the present surface or to the unconformity at the top of the Tuscaloosa. These red iron-oxide stains appear to be primary. Careful attention has been given to both the yellow and red stains and other characteristics of the upper part of the Tuscaloosa, but the writer has seen no clear evidence of subaerial erosion as the origin of the unconformity at the top of the Tuscaloosa, and marine erosion seems the probable cause.

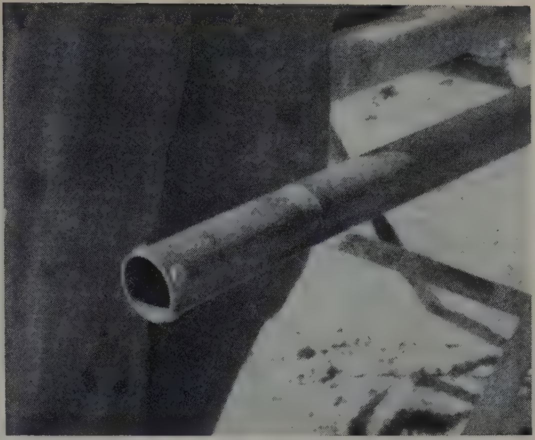

Fig. 3—Bottom end of inner core barrel, showing brass centering studs and 6-in. dry-plug shell.

Pisolitic bauxite, commonly overlain by an extremely hard variety of kaolin known locally as chimney rock, occurs in some places on kaolin at the unconformity above the Tuscaloosa. These occurrences evidently identify truncated kaolin lenses, as sporadic bodies of bauxite are found, in drilling, as much as 30 ft below the unconformity and invariably in the interior of kaolin deposits rather than on their surface. Such buried bauxite is mostly white, but some of it is mottled brick-red and green. The occurrence of the buried bauxite, in thick bodies of kaolin containing no clear evidence of subaerial erosion, strongly indicates that there is no relation between the origin of the bauxite and that of the unconformity. In fact, they appear to be widely different in age—the unconformity being early Tertiary or pre-Tertiary and the bauxite being post-Tertiary and related to weathering.

The only organic remains that have been found in the Tuscaloosa occur in, or immediately above or below, a few of the kaolin deposits. The remains are leaves and grasses that occur in thin beds of lignite, in zones of brown carbonaceous sand and clay as much as 6 ft thick.

Problem of Origin

It has been generally agreed, since the first investigation of Georgia kaolin by Ladd,[2] that the erosion of weathered rocks of the Piedmont furnished the sand and kaolin of the Tuscaloosa formation. The vast amount of haphazardly piled material in the Tuscaloosa indicates rapid erosion and transportation. The debris was evidently deposited in coalescing deltas and alluvial fans along the Cretaceous shore line, and in part redistributed by stream and ocean currents, as shown by an abundance of minor unconformities. The lenses of kaolin

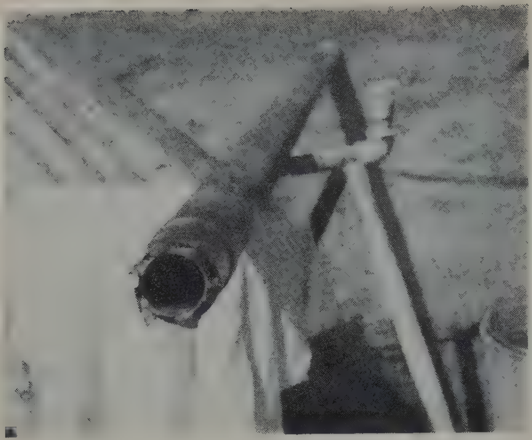

Fig. 4—Bottom end of assembled core barrel, showing general design of the bit including waterways and reaming ring.

were deposited, in pools and lakes of fresh[4] or salt[11] water, nearly isolated from the currents.

Even with scant evidence, a few arguments have developed. Veatch[4] inferred that the weathering that made the debris available for erosion had been in progress since Cambrian time and that only a few feet of red-stained saprolite would have developed in the Piedmont during this time. Thus, he minimized the development and pigmenting power of natural iron oxide in attempting to explain the lack of color in the kaolin deposits. Neumann,[11] in considering the Tuscaloosa kaolin deposits of South Carolina, doubted that long weathering would have developed so little red color, and his objection is sustained by the occurrence of the largest area of mafic crystalline rocks in Georgia, which weather deeply and vividly red, immediately northwest of the kaolin-producing area.[12] Neumann accounted for the lack of red color in the kaolin by assuming that heavy plant growth, which was on a pre-Tuscaloosa Piedmont surface reduced to a peneplain, provided organic acids that leached iron from the rocks as they were weathered.

Since these ideas have developed on the hypothesis that weathering was active continuously in the Piedmont region from Cambrian to Upper Cretaceous, it would be well to examine the underlying premise. The geologic map of Georgia[12] shows dikes of Triassic diabase as well as bodies of Carboniferous granite in the crystalline rocks overlapped by the Tuscaloosa and younger formations. These igneous rocks do not cut the Coastal Plain formations and consequently they must have been a part of the crystalline "complex" eroded to furnish the Coastal Plain sediments. Weathering of genetic importance to the Tuscaloosa, therefore, must have started not earlier than the end of the Triassic, and recent evidence cited by Richards[13] tends to limit further the possible length of time. This evidence involves the discovery of up to 3000 ft of pre-Tuscaloosa and post-Triassic sediments in deep oil tests in eastern North Carolina. All of these beds may be Lower Cretaceous, or the lower 1400 ft may be Jurassic. Approximately equivalent beds are reported by Applin[18] in deep-well cores from other parts of the Coastal Plain, in Florida, Georgia, and Alabama. From this it appears that the Piedmont of Georgia may have been part of a relatively high region undergoing erosion during the Lower Cretaceous if not the Jurassic. The basis for inferring a long pre-Tuscaloosa period of saprolite development, even more extensive than that which produced the Piedmont of today, thus seems to disappear.

Similarly, the basis for disagreement regarding a difference in origin between the soft and hard kaolin seems to disappear in view of conclusive tests by the Bureau of Mines[10] showing no fundamental difference except in particle size, and the occurrence of all grades of semihard kaolin, qualitatively bridging the gap between the extremes. Where a deposit consists of a lower layer of soft kaolin and an upper layer of hard kaolin, it is evident that there first was deposited material of one size and later material of a finer size, with an intervening period of no deposition. A pause in sedimentation seems more likely than continuous sedimentation, with separation according to particle size, as suggested by Smith.[8] Evidence for such a pause is afforded by the usual line of separation between the layers and by laboratory evidence cited by Gongwer,[14] who states that there is no separation of coarser from finer particles during the settling of ground clays suspended in water.

The earlier suggestion by Stull and Bole[15, 16] that silicic acid derived from fullers earth has hardened deposits originally soft is most inadequate in view of the normal stratigraphic relations described herein. The Twiggs clay (fullers earth) overlies the Tuscaloosa in the entire area except where it has been carved out by the modern streams. Consequently, all of the kaolin deposits near the top of the Tuscaloosa have had the same environment probably until the Pleistocene. In spite of this, deposits of both hard and soft kaolin occur in places with a cover of fullers earth, and in other places without it, indicating that neither the superposition of the fullers earth nor its partial removal have influenced the hardness of the kaolin.

Quite recently Bridge[17] has directed attention to the unconformity at the top of the Tuscaloosa in connection with the origin of the bauxite deposits of the Southeast. He believes that the bauxite-containing kaolin deposits are not actually within

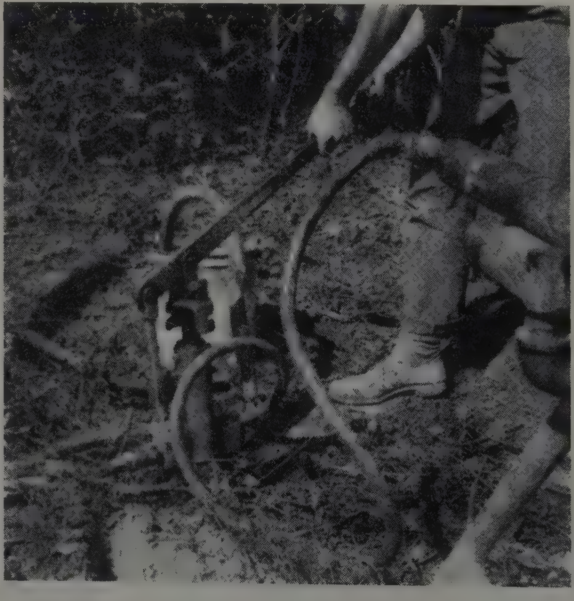

Fig. 5—Hydraulic hand pump used to remove core from inner barrel.

the uppermost part of the Tuscaloosa formation, but instead "rest on and are seemingly a part of" that formation, supposedly being part of a thin veneer of transported materials deposited on an early Tertiary erosion surface "almost contemporaneously with its formation," and largely removed by later erosion. However, all of the evidence seen by the writer in open-cut mines and in cores and cuttings from hundreds of drill holes, many reaching more than 100 ft below the unconformity, shows the uppermost kaolin deposits and associated sediments to be integral parts of the Tuscaloosa. They are not different in age, in origin, or in any characteristics other than the bauxite cores that a very few of them contain and the bleaching or brightening effect of weathering.

Perhaps the prominence of the unconformity has obscured the relation of the bauxite-bearing kaolin bodies to the overlying Twiggs clay and its calcareous beds. It is well known that iron sulphide commonly occurs in many of the kaolin bodies in all parts of the Tuscaloosa and that it has been oxidized only in the uppermost bodies within about 30 ft of the Twiggs clay or the position from which the Twiggs clay has been removed by post-Tertiary erosion. The oxidation appears to be associated strictly with post-Tertiary weathering, as mentioned above. Consequently, any of the uppermost kaolin bodies that were unusually rich in iron sulphide could have developed appreciable amounts of aluminum sulphate through reaction of natural sulphuric acid with the kaolin, and prompt reaction with calcium carbonate brought by ground water from the overlying Twiggs clay could have precipitated aluminum hydroxide. The calcium sulphate produced may have moved away in the ground water, or it may have been deposited with the hydroxide and subsequently leached. Either type of transport would result in some open structure, probably accounting for the occasional total loss of water circulation when the core drill penetrates bauxite.

To the exploration geologist or engineer, the importance of an adequate explanation of origin of the kaolin deposits lies in its application to the problems of discovery. The difference in the grade of the known deposits, plus the occurrence of more than one grade of kaolin in some of them, make it obvious that the most we can expect of a genetic theory is to indicate geologic associations or geographic trends favorable to the occurrence of kaolin lenses in the Tuscaloosa regardless of their specific character. The sketchy explanation thus far advanced provides no assistance.

A new approach might help. If there was no long pre-Tuscaloosa peneplanation and rotting of crystalline rocks, then it is unnecessary to account for deeply red-stained source material. The pell-mell accumulation of coarse sand and clay in the Tuscaloosa reflects a transgressing sea and steep stream gradients, with resulting vigorous erosion that could provide sediment whether or not thorough rotting had occurred. The debris fed to the streams may have been disintegrated rather than leached rock, and further disintegration during transport would have promoted gravity separation of the heavier iron-bearing minerals, from the feldspars and quartz, before oxidation had reached a strongly pigmenting stage. Thus, the sediment that finally reached the lower courses and mouths of the principal streams may have been largely arkosic sands, parts of which were fairly free of iron-bearing minerals.

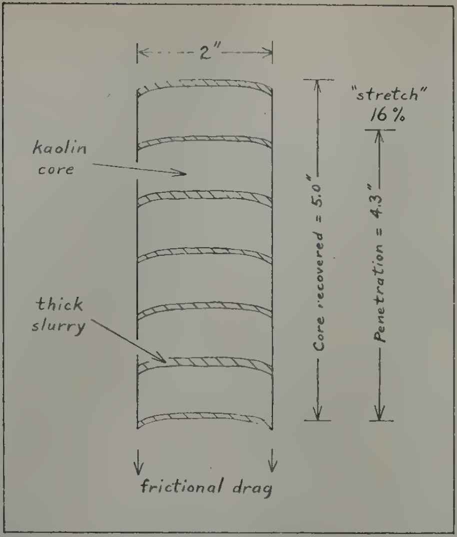

Fig. 6—Transverse shear planes developed in kaolin core.

The final decomposition of the feldspars may have occurred in these alluvial and deltaic deposits, adjacent to the lagoons in which the newly formed kaolin was subsequently deposited during redistribution of the sediments by flood and ocean currents. An origin of this sort would help to explain not only the lack of color but also the presence of sharply angular quartz grains in the smooth, homogeneous kaolin. The angularity of such grains undoubtedly reflects interlocking rock texture, probably an interlocking with feldspar that decayed after reaching the shore line or an adjacent area.

But it is not enough to explain the kind of source material and the mode of transport. To be useful, a genetic theory must take into account the character of the Tuscaloosa shore line and its currents. It must explain the relative abundance of kaolin deposits in the area here considered, from the base of the Tuscaloosa to the highest part exposed. It should indicate, if possible, preferred trends within and from this area, so that prospecting may be focused on favorable parts of the Tuscaloosa that are covered by the younger formations. If any trends are apparent, and if they can be adequately explained, clues may be provided for the discovery of others that are obscure.

Prospecting Methods and Equipment

Until recent years most of the prospecting in the Georgia kaolin country was done by hand boring and the digging of test pits. Various types of screw auger bits were used with ½-in. or ¾-in. pipe in short sections. The bit could be sunk only a few inches at each pass and could be pulled only with the greatest difficulty by three men, unless a tripod were set up after the penetration of as little as 15 ft of kaolin. Samples were contaminated with sand and colored clay from the overburden, and loose, water-bearing sand immediately above the kaolin often choked holes before sufficient depth was reached. A few power drills of auger type had been used but were unsatisfactory because of disturbed and contaminated samples and poor control of footage sampled.

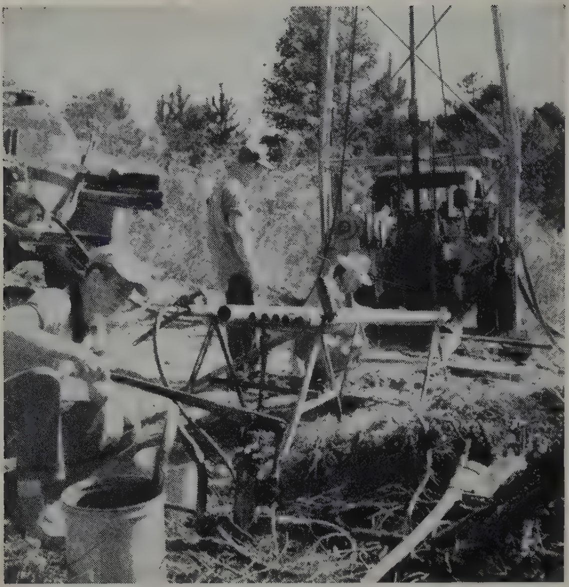

Fig. 7—Kaolin-prospecting outfit, including Failing seismograph drill, supply truck, and arrangement of core barrel for discharging core.

Systematic, deep drilling with high recovery of relatively undisturbed core was started in the latter part of World War II and is rapidly gaining favor. There are now about ten rigs in operation, and the number is likely to increase. Seismograph drills with N rods are used, with one exception, and all operators have adopted a double-tube core barrel, with special bit, constructed so as to permit the core to be lubricated by the water without being greatly reduced in size or washed away. This barrel and bit constitute an ingenious tool peculiarly adapted to the difficulties involved. It was reportedly developed by Lewis C. Lindsay, of the Pennsylvania Drilling Co., who was probably the first to successfully carry on the deep testing of kaolin in the Georgia field, and it is known as the Lindsay barrel. The kaolin operators obtain repairs and replacements at the Kitchens and Harbuck machine shop in Americus, Ga. So far as the writer is aware, the barrel is not available anywhere as an open-stock item.

The inner barrel is of 2-in. tubing and must remain rather closely centered over the bit to insure that the core will enter perfectly at the beginning of the run. This is accomplished with four brass studs, arranged as shown in Fig. 3, which permit little sway of the inner barrel when in operation. The lower 6 in. of the tube is threaded for removal, Fig. 3, and is known as the dry-plug shell. As a core-lifter cannot be used with such plastic material as wet kaolin, the core is held in the barrel by cutting the last 6 in. dry, with increased drill-head pressure, and this lower part of the core (dry plug) must be removed from the shell with a special screw-driven piston. The remainder of the barrel is unloaded as described below.

The bit is of saw-tooth design, with six broad chisels, built of hard alloy, supported by the end of the outer barrel and a 2-in. ring that guides the core into the inner barrel. Spaces between the teeth, the ring, and the outer barrel are the waterways, as shown in Fig. 4. These feed the water directly to the cutting teeth, and a gap of only about 1/16 in. between the ring and the end of the inner barrel permits the water to lubricate the core without remaining in contact with it long enough to erode. The teeth cut ample outside clearance, but the inside clearance is spaced wide to permit the ring to ream the core to the exact size of the inner barrel. After each run the barrel is disassembled, and, after the dry-plug shell is removed, the core is pumped from the inner barrel with a Baker hydraulic hand pump, coupled at the dry-plug connection as shown in Fig. 5. Thus, the barrel discharges from the top, permitting a more orderly handling of core.

Deposits consisting of hard and semihard kaolin give the same difficulty, because of blocking, as do fractured rocks in diamond drilling, but with the kaolin it may result in core loss of 12 to 22 pct of the penetration, even with careful and experienced operation. Soft kaolin, on the other hand, commonly yields excess core measurement owing to closely spaced transverse shear planes, as shown in Fig. 6. These are caused by the adherence of the plastic kaolin to the ring in the bit, thus cutting a "biscuit" each time the ring is filled, and the biscuit rotates with the bit until pushed up into the inner barrel. During the rotation the shear plane below the biscuit admits drilling water, forming a thin sheet of slurry that may contain impurities introduced with the water. These probably would be sand, montmorillonite, and calcite derived from the Twiggs clay in the overburden. The length of the core is extended by these sheets of slurry, and the total "stretch" commonly ranges from 15 to 20 pct of the penetration.

Drilling technique differs somewhat according to the hardness of the kaolin, and the operator must determine for each deposit the most suitable drill-head pressure and rate of water circulation. All parts of the barrel must be washed between runs, and thrust bearings, which are used in the head, must be replaced frequently. Various types of fishtail bits are used in overburden, and the writer has successfully used 4-in. flush-coupled casing where the limestone beds in the Twiggs clay are cavernous. A complete outfit is shown in operation in Fig. 7, and the work is frequently carried on with the aid

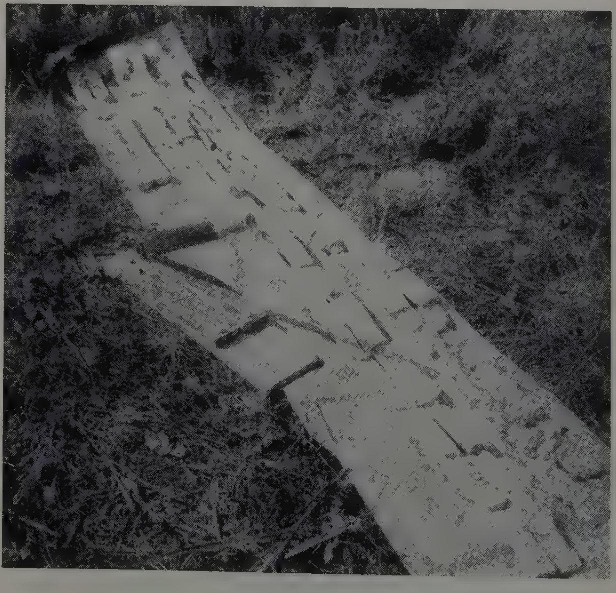

Fig. 8—Fresh split cores of kaolin arranged for inspection and sampling.

of a bulldozer, which cuts benches on steep slopes and plows drillways through the dense growth. The cores recovered in this work may be removed in standard core boxes for laboratory tests, or they may be examined and sampled immediately, see Fig. 8, to reduce bulk and remove obvious contamination.

Outlook for Discovery

Although the size of the kaolin-producing area is quite large, there is a decided difference in outlook between different parts of it. The outlook involves not merely the geologic possibility of the occurrence of kaolin lenses, but also accessibility for prospecting and mining, and possible relation between the depth of overburden and the size and grade of deposits that might be found. Although lenses of kaolin as thick as 50 ft[3] have been found, most of those mined have averaged nearer 20 ft; and although overburden of the order of 100 ft in thickness has been stripped in rare instances, most of it has been less than 60 ft thick, and less than half that amount in the smaller mines. Serious troubles resulting from ground water, particularly in the winter and spring, are encountered in openings in low ground near the swamps and near many of the smaller tributaries.

In view of these general facts as well as the exploration problems involved, the proportionate parts of the area in their relation to prospecting and mining are about as shown in Fig. 2. Of the total of 445 sq miles, about 100 sq miles or 22 pct is in the higher ridges in which all of the overburden, even if kaolin were found at depth, would exceed 75 ft, much of it being substantially more than 100 ft. At the other extreme, the ground that is unfavorably low includes 75 sq miles or 17 pct alluviated swamps and 40 sq miles or 9 pct low, sandy, water-bearing flats generally indicating sandy, easily eroded underlying material.

The remaining 230 sq miles, or 52 pct, of the area is comparatively favorable for prospecting or mining, or both. This part must be reduced still further, however, because many properties have been proved barren by prospecting carried on through the years, and because stream erosion has provided enough exposures on many other properties to show that no large deposits of good grade can be present under thin or moderate overburden. There is no way of closely estimating the extent to which these factors reduce the 52-pct part of the area shown in Fig. 2 as having favorable mining conditions. The full extent and significance of past prospecting are unknown as information is not freely exchanged among the operators, and the extent of dependably indicative natural exposures also is unknown because of a lack of geologic mapping of pertinent features. It may well be, however, that if it were possible to have the advantage of all previous prospecting as well as the results of very detailed areal mapping, it would be found that 25 pct to 50 pct of the 52-pct part of the area actually has no possibility of the discovery, at economic depth, of deposits large enough and good enough to sustain the industry in its present scale of operation.

This outlook need not give rise to pessimism concerning long-range reserves. It merely indicates that the possibilities are not scaled in proportion to the size of the area. Deposits yet to be discovered in this mining area of 445 sq miles will be found mostly in parts of it making up 115 to 175 sq miles in the total. To develop new reserves in these parts will claim an increasing amount of the attention, which, until recent years, has been given almost entirely to mining and milling. With adequate guidance, most of the work can be devoted to relatively favorable ground, an approach now in use in the area; but without guidance, it may be necessary to drill all property that can be leased, another current approach. Regardless of the approach or of the work involved, the industry is never likely to fall back on treating low-grade mixtures of sand and kaolin as long as more deposits of high-grade kaolin can be found.

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Contribution of Geophysical Surveys to the Discovery of Stilfontein Gold Mine in South Africa

by Oscar Weiss

STILFONTEIN and surrounding areas gave negative results in all prospecting attempts since about 1904 when Goerz and Co., a forerunner of Union Corporation Ltd., first drilled in the region and on the farm Stilfontein. Subsequent efforts by African and European Investments Co. and Anglo American Corp. from 1934–37 were completely unsuccessful. Holes drilled by these companies on the farms Rietfontein, Hessie, and Palmietfontein were stopped in the Lower Witwatersrand system without finding payable reefs.

The results of gravimeter, ground magnetometer, and aerial magnetometer surveys are presented in the area, in and around the new Stilfontein Gold Mine, in the Klerksdorp District of the Transvaal in the Union of South Africa. The gravimeter and ground magnetic surveys were completed in 1947–48. The gravity anomalies suggested a block of Upper Witwatersrand quartzites faulted-up between denser rocks of dolomite and Ventersdorp lava. Boreholes S.T. 1, 2, and 3 were drilled in 1948 and, after penetrating the dolomite and Ventersdorp lava, entered Upper Witwatersrand quartzites and intersected the Gold Estates Reef and Vaal Reef. Subsequent drilling delineated the suboutcrop of the Vaal Reef and determined the main structure of this horizon. The gold content of the Vaal Reef was highly payable and shaft sinking is now in progress. The Stilfontein Gold Mining Co. has been formed and, up to date, £5,000,000 capital has been raised for plant, shaft sinking, and development. The payable Vaal Reef horizon extends further south from Stilfontein, the reef occurring at increasing depths. It is possible that at least another mine may be established on these deeper horizons.

The farm Stilfontein 39 is situated in the Klerksdorp District of the Transvaal. This farm and the surrounding areas have been subjected to repeated programs of prospecting dating back to about 1904, when Goerz and Co., the forerunner of Union Corporation, Ltd., first drilled on the farm and in the surrounding areas. From 1934–37, numerous boreholes were drilled on the neighboring farms, Rietfontein, Hessie, and Palmietfontein, by African and European Investment Co. and Anglo American Corp. All these efforts were completely unsuccessful, and nobody suspected any possible gold-bearing area on Stilfontein until about 1947, when a gravity traverse run by B. D. Maree on the main road, showed a negative gravity anomaly. During 1947–48, the writer's organization completed gravimeter and magnetic surveys on Stilfontein and Rietfontein. An area of negative gravity anomalies was mapped, and it was suggested that this could be caused by blocks of Upper Witwatersrand quartzites, of the average density of 2.65, faulted up between dolomites and Ventersdorp lavas, the density of both of these rocks being 2.8.

Fig. 1 shows the geology of the area as mapped by L. T. Nel of the Geological Survey of South Africa. The boreholes on Stilfontein, with the exception of a single hole of Goerz and Co., were drilled after the completion of the geophysical survey. The grouping of the first boreholes S.T. 1, 2, 3, 4, 5, and 6, clearly shows that these were located in the zone of negative gravity anomalies. Fig. 2 shows the gravity anomaly contours.

Fig. 3 shows two sections across the gravity anomaly, together with the corresponding geological sections as obtained from boreholes. The close relation between gravity values and thickness of dolomite plus Ventersdorp lavas is striking. The two latter formations having the same density (2.8), act as one mass as far as gravity effects are concerned.

Fig. 4 shows the magnetic anomalies observed on the ground by vertical magnetometers and at 500 ft above mean ground level by aerial magnetometer. The latter are anomalies of the total magnetic intensity. The magnetic results show that, on the farm Rietfontein and on the northern half of Stilfontein, Lower Witwatersrand magnetic shales were to be expected at relatively shallow depths. As the strongly

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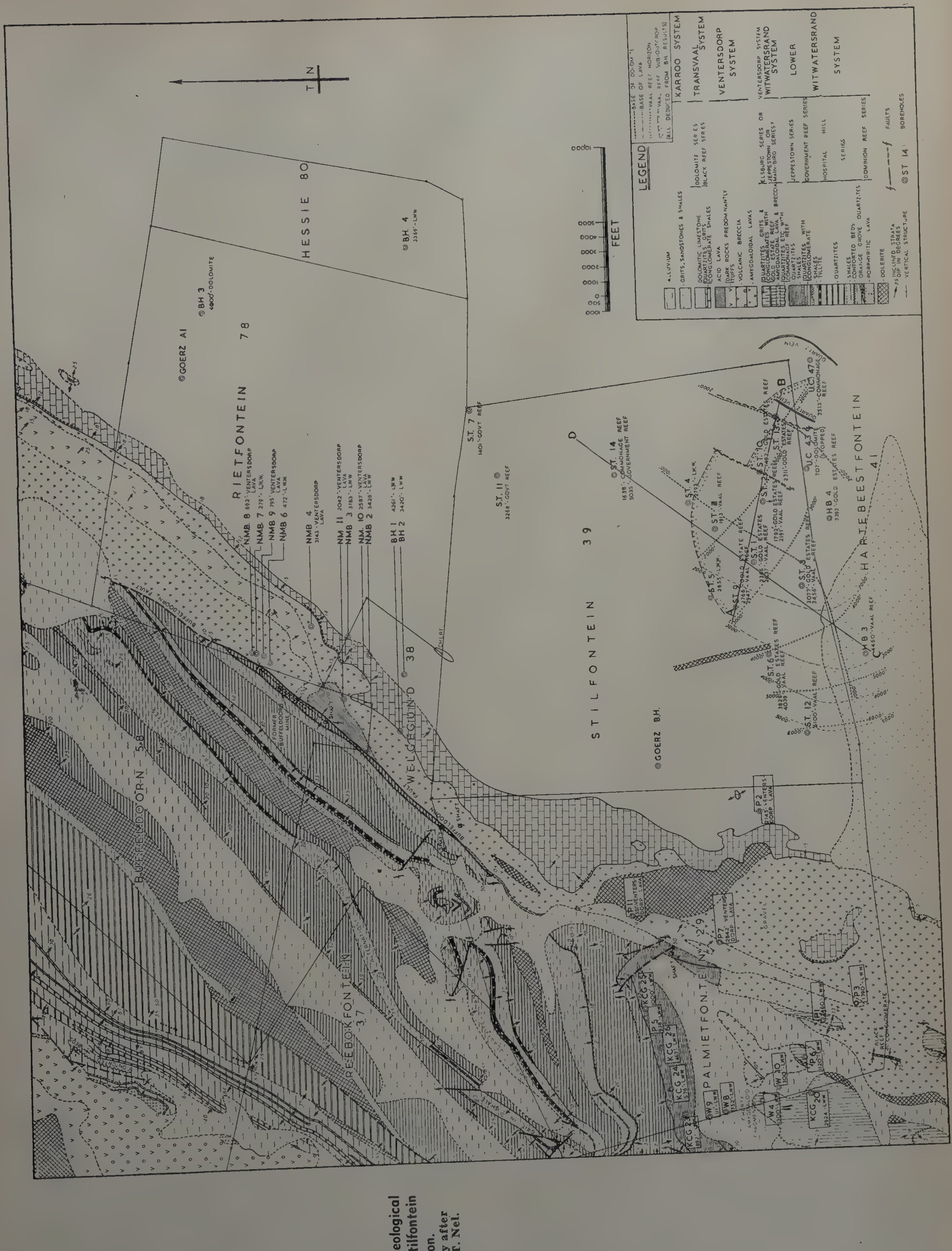
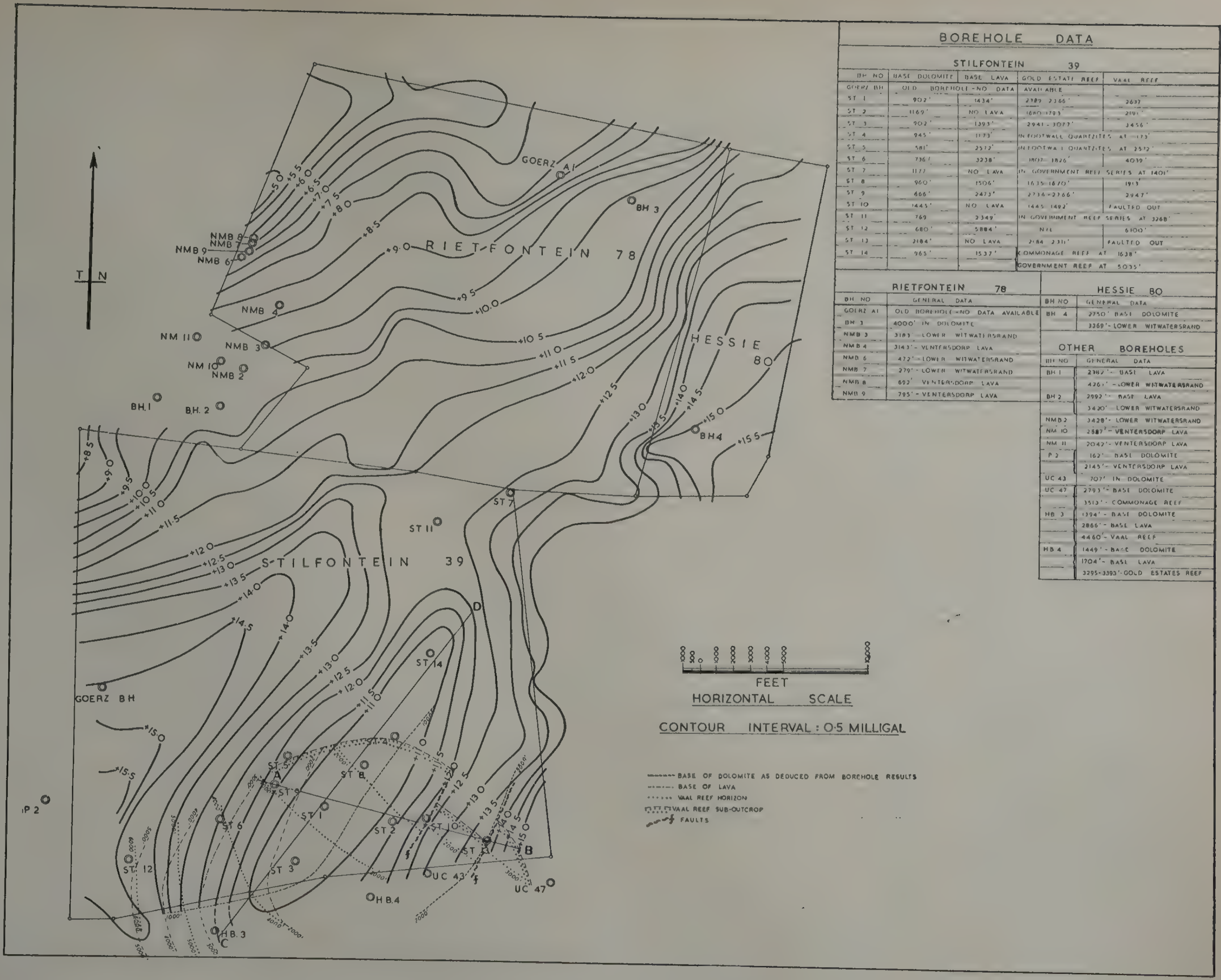


Fig. 1—Geological plan of Stilfontein region. Geology after Louis T. Nel.



BOREHOLE DATA

STILFONTEIN 39

BH NO	BASE DOLOMITE	BASE LAVA	GOLD ESTATE REEF	VAAL REEF
GOERZ BH	OLD BOREHOLE - NO DATA AVAILABLE			
ST 1	902'	1434'	2289 2366'	2637
ST 2	1169'	NO LAVA	1680-1793'	2191'
ST 3	902'	1393'	2941 - 3077'	3456'
ST 4	945'	1173'	IN FOOTWALL QUARTZITES AT 1173'	
ST 5	581'	2572'	IN FOOTWALL QUARTZITES AT 2572'	
ST 6	736'	3238'	1807 1826'	4039'
ST 7	1177	NO LAVA	IN GOVERNMENT REEF SERIES AT 1401'	
ST 8	960'	1506'	1635 1870'	1913'
ST 9	666'	2473'	2736-2766'	2947'
ST 10	1445'	NO LAVA	1445 1492'	FAULTED OUT
ST 11	769	2349'	IN GOVERNMENT REEF SERIES AT 3268'	
ST 12	680'	5884'	NIL	6100'
ST 13	2184'	NO LAVA	2184 2311'	FAULTED OUT
ST 14	963'	1537'	COMMONAGE REEF AT 1638' GOVERNMENT REEF AT 5035'	

RIETFONTEIN 78

BH NO	GENERAL DATA
GOERZ A1	OLD BOREHOLE - NO DATA AVAILABLE
BH 3	4000' IN DOLOMITE
NMB 3	3183' - LOWER WITWATERSRAND
NMB 4	3143' - VENTERSDORP LAVA
NMB 6	472' - LOWER WITWATERSRAND
NMB 7	279' - LOWER WITWATERSRAND
NMB 8	692' VENTERSDORP LAVA
NMB 9	795' - VENTERSDORP LAVA

HESSIE 80

BH NO	GENERAL DATA
BH 4	2750' BASE DOLOMITE 3369' - LOWER WITWATERSRAND

OTHER BOREHOLES

BH NO	GENERAL DATA
BH 1	2382' - BASE LAVA 4261' - LOWER WITWATERSRAND
BH 2	2992' - BASE LAVA 3420' LOWER WITWATERSRAND
NMB 2	3428' - LOWER WITWATERSRAND
NM 10	2587' - VENTERSDORP LAVA
NM 11	2042' - VENTERSDORP LAVA
P 2	162' - BASE DOLOMITE 2145' - VENTERSDORP LAVA
UC 43	707' IN DOLOMITE
UC 47	2793' - BASE DOLOMITE 3513' - COMMONAGE REEF
HB 3	1394' - BASE DOLOMITE 2866' - BASE LAVA 4460' - VAAL REEF
HB 4	1449' - BASE DOLOMITE 1704' - BASE LAVA 3295-3393' - GOLD ESTATES REEF

Fig. 2—Gravimeter survey of Stilfontein.

magnetic shales of the lower Witwatersrand system[1] are in the Hospital Hill series and in the Government Reef series, below any known gold-bearing pay horizons, the existence of magnetic anomalies beneath the dolomite and Ventersdorp systems was an indication of the probable absence of pay horizons. At the same time the lack of magnetic anomalies in the southern portion of Stilfontein suggested the absence of major upthrows of the Lower Witwatersrand magnetic shales.

The geological plan, see Fig. 1, shows that the oldest formation of the area, the contorted beds near the base of the Witwatersrand system, outcrop in the northwest corner of the plan. Moving in a southeasterly direction from this corner, the Witwatersrand system is faulted up repeatedly by strike faults as shown by the repetition of Government Reef shales and tillites, so that, in this direction, the outcrops do not show horizons higher than the Government Reef series in the Witwatersrand system. This system is overlain by Ventersdorp lavas and sediments and by black reefs and dolomites of the Transvaal system. This dolomite covers the entire area of

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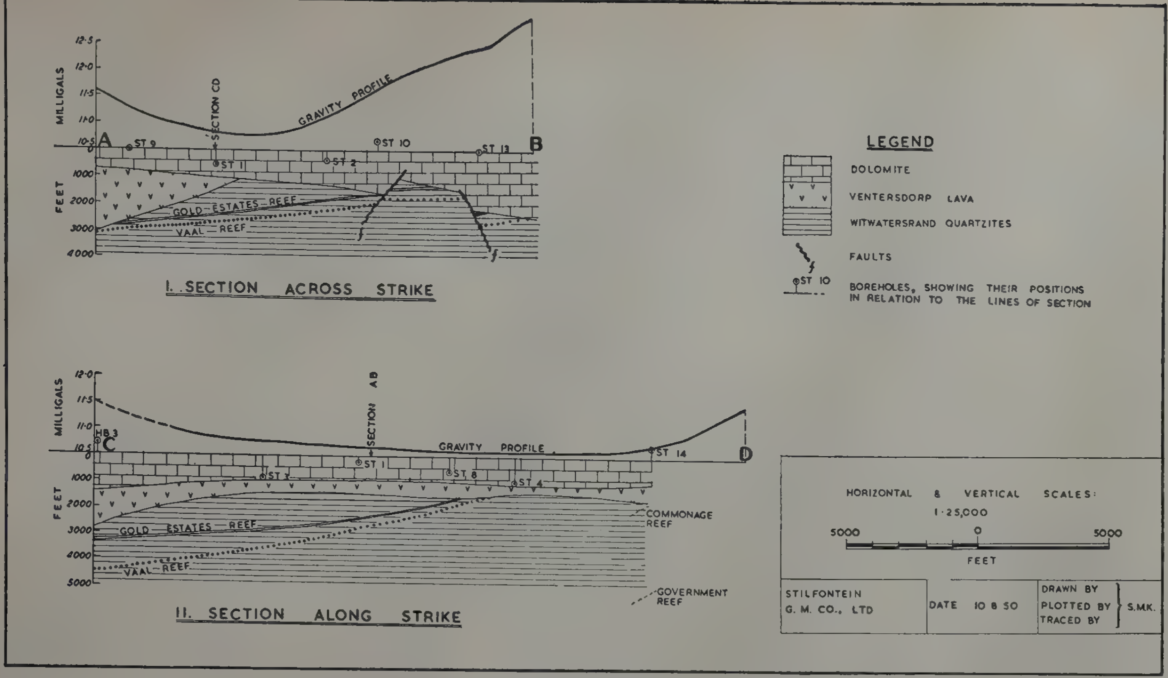

Fig. 3—Geophysical survey Stilfontein Gold Mine, Sections AB and CD.

Stilfontein with the Ventersdorp lava and black reef outcropping in a small portion of the northwest corner of this farm.

Starting again from the northwest corner of the geological plan and moving southward, we again find faulted repetitions of the Government Reef series, but the Witwatersrand system builds up to the Jeppestown series and even higher in the extreme south corner of the plan, where the Elsburg series is outcropping. As the Main Reef horizon of the Rand proper, near Johannesburg, occurs above the Jeppestown shales, exploration drilling by Anglo American Corp. (1934–37) in this part of the plan, on the farm Palmietfontein, was intended to intersect the Main Reef horizon. Unfortunately, in spite of systematic drilling, no pay reefs of any kind were found and all the holes stopped in Lower Witwatersrand beds (LWW).

On the farm Rietfontein, all the NMB holes were drilled by African and European Investment Co.'s subsidiary (New Machavie Mine) and by Anglo American Corp., during 1934-37, boreholes (B.H.) 1, 2, 3. In addition, Goerz A.1 hole was drilled in about 1903. The purpose of all the NMB holes and of boreholes 1 and 2 was to find the extension of the old Buffelsdoorn mine, which worked rich conglomerate beds correlated with Government Reefs. None of this drilling was successful.

Boreholes Goerz A.1 and Anglo American Corp.'s boreholes 3 and 4 (on farm Hessie) were located to find the Main Reef Horizon. These holes were also failures.

It seems that the system of up-throw faults continues to the southeast in the areas tested by these holes. It is interesting to note that the farm Stilfontein was completely ignored by the companies who sponsored the drilling programs.

The gravity anomaly plan shows that, starting from the Ventersdorp lava outcrops in the northwest corners of Stilfontein and Rietfontein, the gravity anomalies increase as the Ventersdorp lava, black reef and dolomite is piling up, but the southeastern half of Stilfontein is an exception. Here, after reaching the value of 14 milligals, the gravity anomaly decreases to a minimum of 11 milligals and rises again to 15 milligals in the extreme southeast corner of the farm. There is thus a negative anomaly of about 3 to 4 milligals and the total thickness of the Ventersdorp lava and dolomite is less in this zone of negative anomalies than in the surrounding areas. Furthermore, as the density of Lower Witwatersrand magnetic shales is also about 2.8, no negative gravity anomalies could occur if the lava and dolomite were underlain by such shales. It was therefore concluded that the negative gravity anomaly was caused either by Upper Witwatersrand quartzites (density 2.65) or by a granite mass (density 2.63). As the aim of the prospecting was to test by drilling Upper Witwatersrand horizons

Table I. Vaal Reef Intersections in Boreholes of Stilfontein Gold Mining Co.

STILFONTEIN 39				
BOREHOLE NO.	THICKNESS OF VAAL REEF	GOLD VALUE PER INCH	TOTAL VALUE INCH-DWTS	REMARKS
S.T. 1	5·75 INCHES 20·75 " 5·00	17·5 DWTS. 3·1 " 13·8 "	101 IN-DWTS. 65 " 69	 1st DEFLECTION 2nd "
S.T. 2	3·00 " 12·25 "	31·2 " 20·9 "	94 " 256 "	 DEFLECTION
S.T. 3	7·25 "	103·4 "	750 "	
S.T. 4	BEYOND VAAL REEF HORIZON			
S.T. 5	BEYOND VAAL REEF HORIZON			
S.T. 6	4·70 INCHES 5·75 "	34·6 DWTS. 22·5 "	162 IN-DWTS 129 "	 DEFLECTION
S.T. 7	BEYOND VAAL REEF HORIZON			
S.T. 8	7·50 INCHES	80·9 DWTS	607 IN-DWTS	
S.T. 9	9 50 "	149 4	1419 "	
S.T. 10	VAAL REEF FAULTED OUT			
S.T. 11	BEYOND VAAL REEF HORIZON			
S.T 12	13·40 INCHES 18·60 " 16·80 "	5·0 DWTS 2·85 " 34·93	67 IN-DWTS 53 585 "	 1st DEFLECTION 2nd "
S.T. 13	VAAL REEF FAULTED OUT			
S.T. 14	BEYOND VAAL REEF HORIZON			
HARTEBEESTFONTEIN 41				
H.B. 3	5 80 INCHES	615 DWTS	308 IN-DWTS	
H.B. 4	VAAL REEF NOT INTERSECTED DUE TO DYKE AT 3604'			
U.C. 43	STOPPED IN DOLOMITE			
U.C. 47	BEYOND VAAL REEF HORIZON			

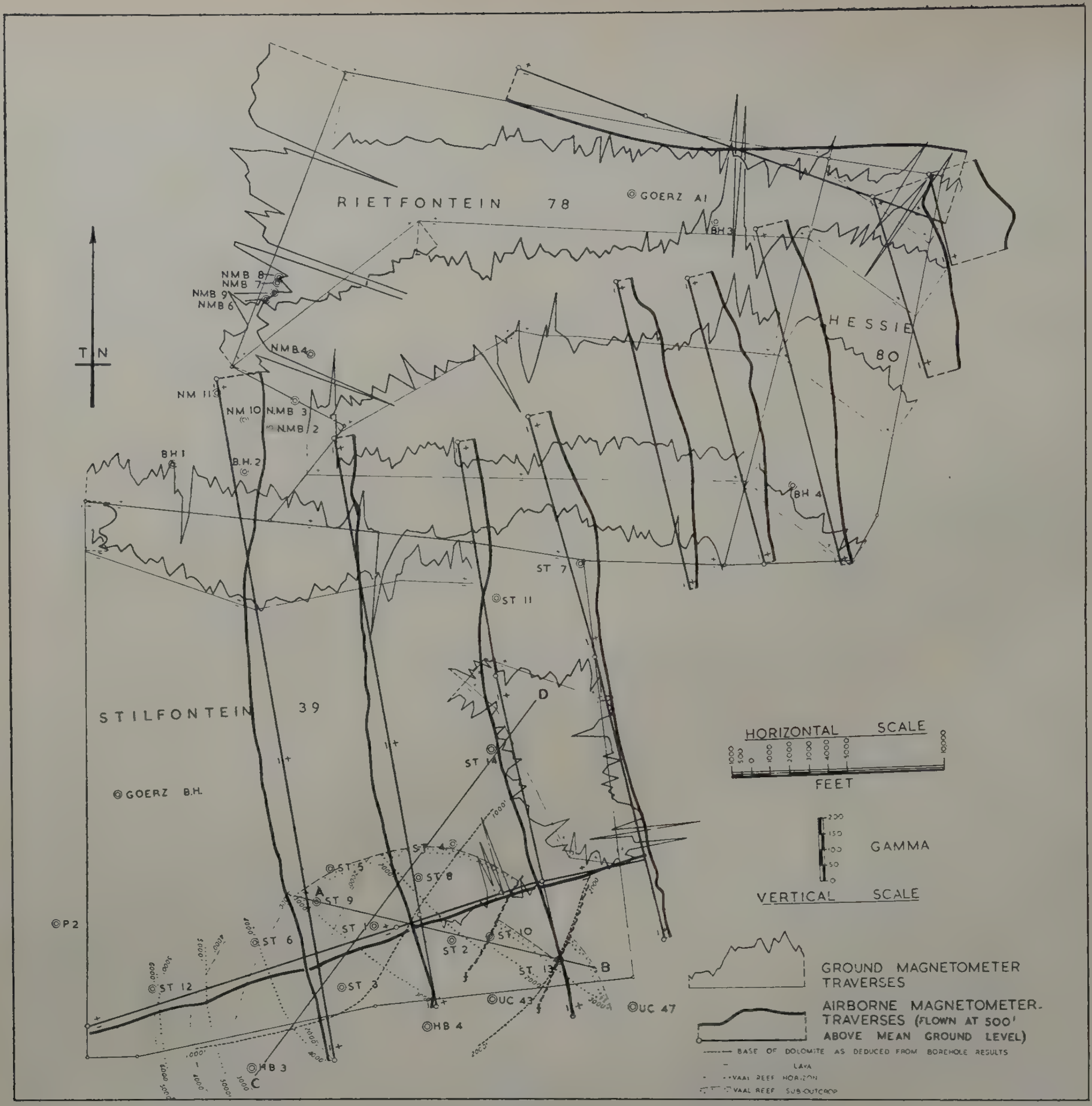

Fig. 4—Magnetic survey of Stilfontein.

for reefs, the obvious sites for such tests were located in the minimum zone of the negative anomaly. The grouping of the holes S.T. 1, 2, 3, 4, 5, 6, 8, 9, and 14 in the negative anomaly area proves the practical following up of the above ideas. All these holes intersected Upper Witwatersrand beds. Boreholes S.T. 7 and S.T. 11 were located outside the negative gravity anomaly, and these holes went into Government Reef, i.e., into Lower Witwatersrand beds. It should be mentioned that the last figure of footage next to any borehole gives the depth at which the hole was stopped.

Table I gives the widths and gold values intersected on Stilfontein.

The approximate contours of the Vaal Reef are shown in Figs. 1, 2, and 3; and, as can be seen, the reef extends southwards, although at increasing depths. It is very likely that at least another gold mine will be established in due course in the area south of Stilfontein.

On the completion of these holes, the Stilfontein Gold Mining Co. was formed with a capital of £3,500,000. To date, £5,000,000 has been raised, by means of share issues and loans, for shaft sinking, development and plant. The total share capital is 14 million 5-shilling shares, of which 11 million shares have been issued and the present market value of these is about £1 per share. A convertible unsecured loan has been obtained for £2,000,000 from London bankers and financial institutions.

Assuming of the order of 30 to 40 million tons potential ore reserves, we can expect a mine with the capacity of about 100,000 tons of ore per month and with a lifetime of about 30 to 35 years. The average grade, as a sheer guess, may be of the order of about 6 to 8 dwt per ton. Average cost per ton, allowing for the increasing costs of present times, should be about 3 dwt per ton, i.e. about 35 shillings per ton. Thus an average profit of 3 to 5 dwts, i.e., 36 to 60 shillings per ton may be expected. Taxation formula payable to the Government has not yet been settled.

Acknowledgment

The writer wishes to thank Jack Scott and Union Corp. Ltd., for permission to publish the technical data of this paper.

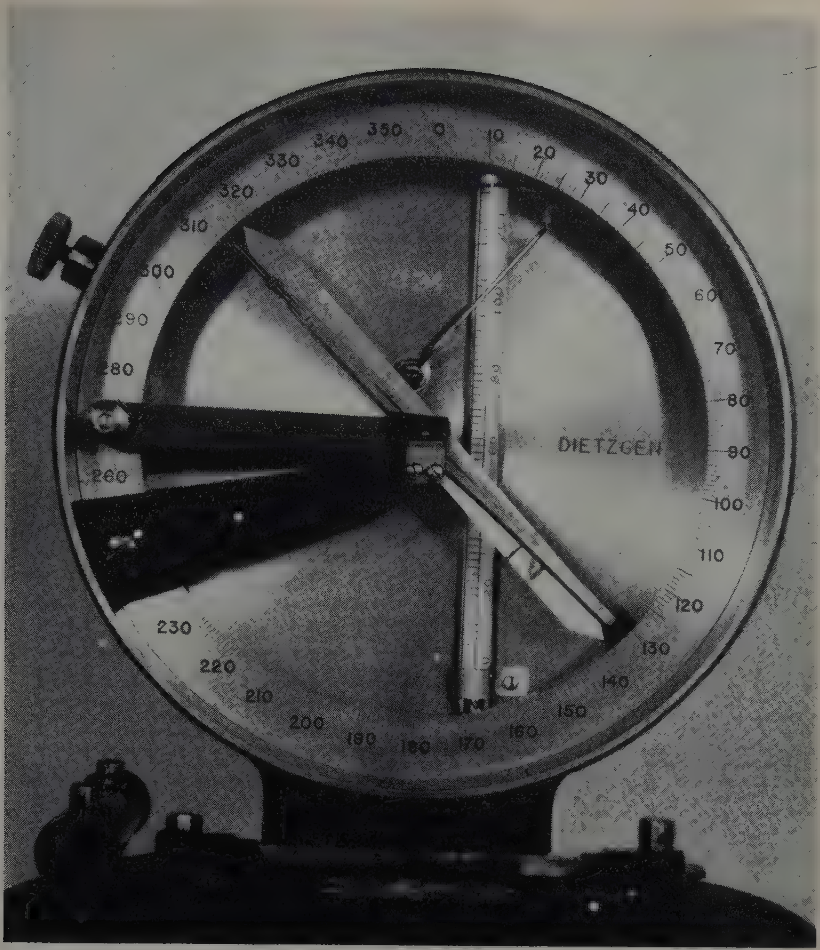

Fig. 1—Hotchkiss Superdip needle system.

The Hotchkiss Superdip As a Vertical Intensity Magnetometer

* * *

by W. A. Longacre

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IN the geophysical exploration program of the Cleveland-Cliffs Iron Co. on the Marquette Range of Northern Michigan, the vertical intensity magnetometer has been used to obtain magnetic anomaly maps of large areas. A network of magnetic base stations has been established and the relative vertical intensity at each station measured carefully with respect to a prime base.

In the search for the relatively nonmagnetic soft red hematite ore, the magnetic exploration is indirect, involving the study of dikes, faults, contacts, and other structural features. It was found that the magnetometer, adjusted to a scale value of approximately 40 γ per scale division, was satisfactory for this type of exploration, although many magnetic anomalies were encountered that required the use of auxiliary magnets.

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The magnetic exploration for the harder type of ore, a mixture of specular hematite and magnetite, is much more direct. This ore is usually very magnetic, although large variations in the magnetite content do cause correspondingly large changes in the magnetic susceptibility. Some additional complicating factors are often present; the hard ore is frequently associated with magnetic beds of an iron silicate complex, occasionally with a conglomerate of iron formation material that may be quite magnetic, and almost always with sills and dikes of diorite. The latter material, while usually much less magnetic than the hard ore, is found to range in magnetic properties from nearly nonmagnetic to strongly magnetic.

The magnetic anomalies encountered in hard ore exploration are often very large and occasionally of such a complicated nature that interpretation becomes difficult. Since the ordinary vertical intensity magnetometer does not have the necessary range to permit measurement of the anomalies, the Hotchkiss Superdip was adopted for this phase of the exploration. It was considered desirable to continue mapping changes in the vertical component of the earth's magnetic field, so the Superdip was used with magnetic east-west orientation. However, since the anomalies measured by the Superdip were expressed in degrees, and the anomalies resolved by the magnetometer expressed in gammas, it soon became apparent that for better mapping and clearer interpretation it would be desirable to calibrate the degree scale of the Superdip in terms of gammas.

In a recent paper,[1] H. L. James of the U. S. Geological Survey described the use of the Hotchkiss Superdip oriented normal to the magnetic meridian and compared its performance with both Askania and Wolfson vertical intensity magnetometers. It was demonstrated clearly that Superdip data, taken with east-west orientation, correlate better with magnetometer data than does the data taken with the Superdip in the conventional magnetic meridian orientation.

Fig. 1 is a photograph and Fig. 2 a schematic representation of a carefully designed Hotchkiss Superdip needle system. The magnetic needle is pivoted at its center of gravity, and the magnetic poles are symmetrically positioned on the longitudinal axis of the needle. With the counterweight removed, the nonmagnetic needle is pivoted at its center of gravity. The two needles are clamped together firmly, the angle between them being adjustable.

The magnetized needle is of pole-strength m, magnetic length $2\,l$, and magnetic moment $M = 2\,m\,l$. The counterweight is at distance a from the pivot and has weight W. The angle between the two needles is Σ. The reading angle θ is measured clockwise from the upper vertical line through the pivot to the N-end of the magnetized needle. The vertical component of the earth's field is V. It is assumed that any self-consistent system of units is used.

The equilibrium equation of the instrument is given by

$$mV\,(2\,l\sin\theta) = Wa\sin(\theta-\Sigma)$$

or

$$MV\sin\theta = Wa\sin(\theta-\Sigma) \qquad [1]$$

$$MV\sin\theta = Wa\,(\sin\theta\cos\Sigma-\cos\theta\sin\Sigma)$$

$$\frac{MV}{Wa} = \cos\Sigma-\sin\Sigma\cot\theta$$

$$\cot\theta = \cot\Sigma-\frac{MV}{Wa\sin\Sigma} \qquad [2]$$

In eq 2, the reading angle θ is expressed implicitly as a function of the vertical intensity V.

The sensitivity of the Superdip is given by

$$S = \frac{d\theta}{dV} \qquad \text{Differentiating eq 2,}$$

$$-\csc^2\theta\,\frac{d\theta}{dV} = 0-\frac{M}{Wa\sin\Sigma}$$

$$S = \frac{d\theta}{dV} = \frac{M\sin^2\theta}{Wa\sin\Sigma} \qquad [3]$$

The sensitivity is large when $\sin\Sigma$ is small, but also depends on the reading angle θ. Eqs 2 and 3 may be combined to eliminate the term $Wa\sin\Sigma$, leading to

$$S = \frac{\sin^2\theta\,(\cot\Sigma-\cot\theta)}{V} \qquad [4]$$

The reading angle θ for which the sensitivity is maximum can be found by differentiating eq 4 and equating $\frac{ds}{d\theta}$ to zero. Thus,

$$\frac{ds}{d\theta} = \frac{1}{V}\,[\sin^2\theta\,(\csc^2\theta) + (\cot\Sigma-\cot\theta)\,(2\sin\theta\cos\theta)] = 0$$

$$\frac{1}{V}\,[1+\cot\Sigma\sin 2\,\theta-2\cos^2\theta] = 0$$

$$\frac{1}{V}\,[1+\cot\Sigma\sin 2\,\theta-1-\cos 2\,\theta] = 0$$

$$\frac{1}{V}\,[\cot\Sigma\sin 2\,\theta-\cos 2\,\theta] = 0$$

$$\cot\Sigma\sin 2\,\theta = \cos 2\,\theta$$

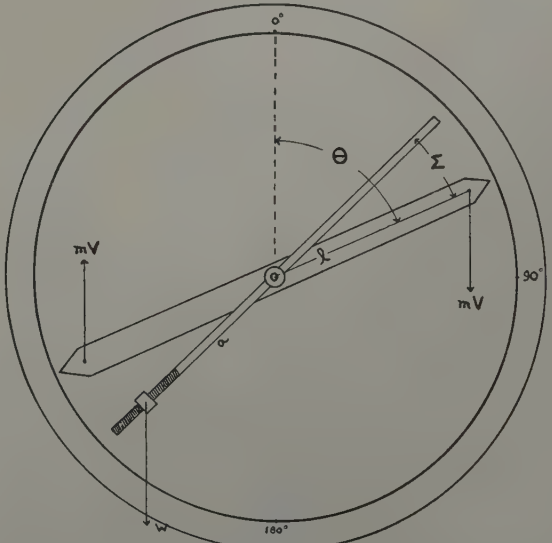

Fig. 2—Schematic representation of Hotchkiss Superdip needle system.

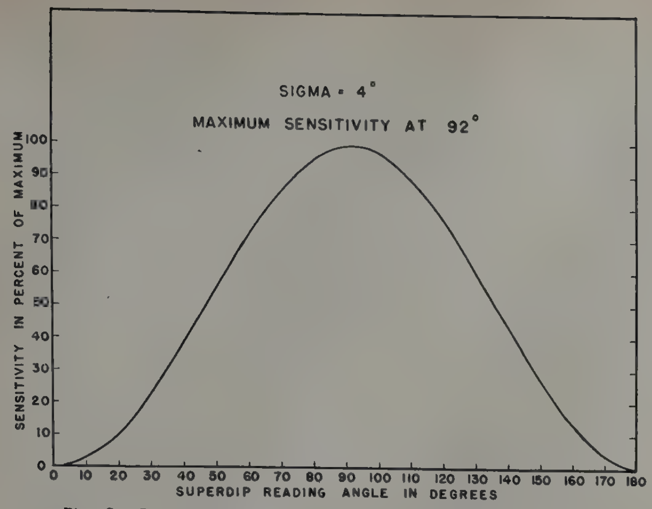

Fig. 3—Superdip sensitivity, a function of the reading θ.

$$\cot \Sigma \tan 2\,\theta = 1$$

$$\tan 2\,\theta = \tan \Sigma = \tan\,(180^\circ + \Sigma) = \tan\,(360^\circ + \Sigma)$$

$$2\,\theta = \Sigma = 180 + \Sigma = 360 + \Sigma$$

$$\theta = \frac{\Sigma}{2} \text{ (minimum)}$$

$$\theta = 90 + \frac{\Sigma}{2} \text{ (maximum)}$$

$$\theta = 180 + \frac{\Sigma}{2} \text{ (minimum)}$$

When $\theta = 90^\circ + \frac{\Sigma}{2}$, the sensitivity is a maximum for any given field V and any given Σ.

The sensitivity of the Superdip as a function of the reading angle is given in Fig. 3.

Superdip Calibration

From eq 1

$$MV \sin \theta = Wa \sin\,(\theta - \Sigma)$$

$$V = \frac{Wa}{M}\,\frac{\sin\,(\theta - \Sigma)}{\sin \theta}$$

If it is assumed that the ratio $\frac{Wa}{M}$ remains constant for a given Superdip, then several methods of calibration are possible.

1—If magnetic base stations of known absolute vertical intensity are available, readings taken at these stations will permit the determination of the most probable value of the ratio $\frac{Wa}{M}$, and a calibration curve for the instrument can be obtained by simple calculation.

2—If a suitable Helmholtz coil is designed and calibrated, data could be taken at a station leading to the calculation of the absolute vertical intensity at the station as well as the value of the ratio $\frac{Wa}{M}$.

3—If a number of stations are located across an area of large anomaly and the relative vertical intensity of each determined carefully with a magnetometer, these stations could be occupied by the Superdip, and data obtained leading to a calibration curve. The calculations will be more dependable if the temperature of the Superdip is nearly constant and near the standard temperature to which all readings are corrected and the diurnal variation of the earth's field not appreciable during the experiment.

Base stations of known absolute vertical magnetic intensity were not available within a convenient distance. Since longer time intervals between stations would likely result in large temperature changes, and also large diurnal change in the magnetic field, it was decided that method 1 was not practicable. A suitable Helmholtz coil was not available, and since the design and construction of the coil promised to be a formidable undertaking with the facilities available, method 2 was not tried. Therefore method 3 was used in the calibration of the Superdip and forms the basis for this paper.

Calibration, Method 3

One of the magnetometer base stations was known to be near an area of high magnetic anomaly. In a traverse, across the anomalous area, seven stations were located so as to give a wide range of relative vertical intensity, the first being about 1000 γ higher than the base, the seventh being over 17,000 γ higher than the base value. At each station, wooden stakes were driven firmly into the ground in a triangular pattern; then a hole was drilled in the top of each stake to provide a firm and stationary location for each leg of the instrument tripod. The Superdip tripod legs were adjusted in length so that the Superdip needle system would occupy the same position in space at a station as had been occupied by the magnetometer system. This was important because of the probability of there being large horizontal and vertical gradients of magnetic intensity in a strongly magnetic area.

The anomaly in magnetic vertical intensity relative to the base was measured carefully at each station with a Ruska magnetometer and checked with an Askania magnetometer. Accurately calibrated auxiliary magnets were used at the higher level stations.

The Superdip, with Σ angle at 4°, was then used in occupying the base and the seven related stations. The circuit was made twice with multiple readings being taken at each station. Individual readings were taken as the weighted average of the end points of the first swing, the second swing, and the third swing; the end point of the second or reverse swing being doubly weighted. All readings were corrected approximately for temperature changes and for diurnal variations of the magnetic field.

Superdip calibration data for June 1949 are given in table I.

These data, by substitution in eq 1, lead to the following equations:

$$V_4 + 0 \quad = \frac{Wa}{M}\,\frac{\sin\,(\;58.3 - 4)^\circ}{\sin 58.3^\circ} = 0.95448\,\frac{Wa}{M}$$

Table I. Calibration Data, June 1949

Station No.	4	4-1	4-2	4-3	4-4	4-5	4-6	4-7
Anomaly in Gammas, Relative to Sta. 4	0	1009	1971	3385	6914	10,317	13,820	17,474
Corrected Superdip Reading, deg.	58.3	69.6	82.1	101.8	137.3	151.9	159.6	164.4

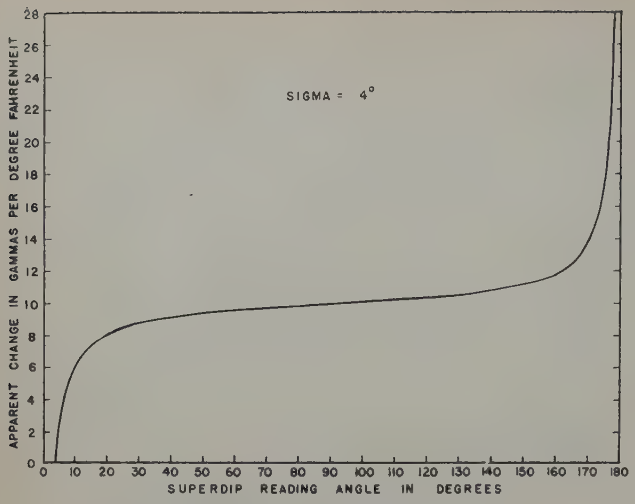

Fig. 4—Calibration curve, Superdip No. 334.

$$V_4 + 1009 = \frac{Wa}{M}\,\frac{\sin\,(69.6 - 4)^\circ}{\sin 69.6^\circ} = 0.97162\,\frac{Wa}{M}$$

$$V_4 + 1971 = \frac{Wa}{M}\,\frac{\sin\,(82.1 - 4)^\circ}{\sin 82.1^\circ} = 0.98789\,\frac{Wa}{M}$$

$$V_4 + 3385 = \frac{Wa}{M}\,\frac{\sin\,(101.8 - 4)^\circ}{\sin 101.8^\circ} = 1.01214\,\frac{Wa}{M}$$

$$V_4 + 6914 = \frac{Wa}{M}\,\frac{\sin\,(137.3 - 4)^\circ}{\sin 137.3^\circ} = 1.07316\,\frac{Wa}{M}$$

$$V_4 + 10{,}317 = \frac{Wa}{M}\,\frac{\sin\,(151.9 - 4)^\circ}{\sin 151.9^\circ} = 1.12821\,\frac{Wa}{M}$$

$$V_4 + 13{,}820 = \frac{Wa}{M}\,\frac{\sin\,(159.6 - 4)^\circ}{\sin 159.6^\circ} = 1.18513\,\frac{Wa}{M}$$

$$V_4 + 17{,}474 = \frac{Wa}{M}\,\frac{\sin\,(164.4 - 4)^\circ}{\sin 164.4^\circ} = 1.24740\,\frac{Wa}{M}$$

By application of the method of least squares the most probable value of V_4 was found to be 57,100 γ and the most probable value of $\frac{Wa}{M}$ found to be 59,780 γ.

Base 4 was known to be about 320 γ higher in magnetic level than the prime base, which has been assigned an arbitrary value of 1000 γ. The calculated absolute vertical intensity at the prime base, about 56,780 γ, was thus a byproduct of the Superdip calibration. The normal value for the area in 1949, estimated from U. S. Coast and Geodetic Survey charts, was about 58,300 γ, indicating that the prime base had been located in an area where the magnetic vertical intensity was some 1500 γ below the normal value.

The experiment was repeated at the beginning of the 1950 field season. The method was the same as before except for minor changes in technique aimed at improving temperature corrections for the Superdip and improving the treatment of diurnal variations. The results were in fair agreement with the ones obtained previously. The comparison of measured values for 1949 and 1950 is given in Table II.

Table II. Comparison of Measured Values, 1949 and 1950

	1949	1950
V_4, γ	57,100	56,560
$\frac{Wa}{M}$, γ	59,780	59,360
Prime base, γ	56,780	56,240

Prime Base Absolute Vertical Field

Shortly after the 1950 calibration experiment, an absolute magnetometer of the flux-gate type was obtained. This instrument, manufactured by the General Electric Co. for the Navy Bureau of Ordnance, has been used in checking the degaussing of ships and is known as the Mark V, Model 4 portable magnetometer. It is supposedly accurate to about 100 γ. By careful orientation of the permeable unit, it is possible to measure any component of the earth's magnetic field.

Repeated trials at the prime base with the absolute magnetometer gave an average value for the absolute vertical intensity of 56,100 γ, with the individual trials ranging from 56,000 to 56,300 γ.

Superdip Calibration Curve

Magnetometer station values had been obtained by measuring the difference in magnetic vertical intensity between a field station and the prime base, and adding algebraically this difference in gammas to 1000. Thus, to a station 200 γ higher in magnetic level than that of the prime base would be assigned a value of 1200 γ. It was intended that Superdip data should lead to approximately the same station value as would be obtained by magnetometer observations.

The absolute vertical intensity at any station is expressed by eq 1 as

$$V = \frac{Wa}{M}\,\frac{\sin\,(\theta - \Sigma)}{\sin\theta}$$

For the Superdip being used, $\Sigma = 4^\circ$, and $\frac{Wa}{M} = 59{,}360$ γ. Thus,

$$V = 59{,}360\,\frac{\sin\,(\theta - 4)^\circ}{\sin\theta}$$

The anomaly relative to the prime base is given by:

$$\Delta V = V - 56{,}240 = 59{,}360\,\frac{\sin\,(\theta - 4)^\circ}{\sin\theta} - 56{,}240$$

The station value in gammas is then given by:

$$\text{Station value} = \Delta V + 1000 = 59{,}360\,\frac{\sin\,(\theta - 4)^\circ}{\sin\theta} - 56{,}240 + 1000$$

$$\text{Station value} = 59{,}360\,\frac{\sin\,(\theta - 4)^\circ}{\sin\theta} - 55{,}240$$

In the latter formula the only troublesome calculation is the evaluation of the trigonometric ratio. This ratio was determined for each integral degree from 4° to 179°. As long as the Σ angle is held unchanged, this calculation need not be repeated for subsequent recalibration of the instrument. The station value corresponding to a Superdip reading angle is obtained by multiplying the corresponding trigonometric ratio by the instrument constant $\frac{Wa}{M}$, and subtracting from this result the absolute vertical intensity at the prime base less 1000 γ.

To facilitate the use of the calibration data, a set of curves was plotted showing the station value as ordinate and the reading angle of the Superdip as abscissa, see Fig. 4. The extremes of the calibration curve were plotted on semilogarithmic paper; other intervals were plotted with simple and convenient ordinate scales.

Temperature and Superdip Readings

To investigate the theoretical effect of temperature changes on the Superdip, reference is made to eq 1

$$V = \frac{Wa}{M}\,\frac{\sin(\theta-\Sigma)}{\sin\theta}$$

At a station where V is fixed, a change in temperature would produce a change in the term $\frac{Wa}{M}$ and a consequent change in the reading angle θ. The differentials of the terms of the above equation are found, on the assumption that V and Σ remain constant. Thus:

$$O = \frac{Wa}{M}\,d\,\frac{\sin(\theta-\Sigma)}{\sin\theta} + \frac{\sin(\theta-\Sigma)}{\sin\theta}\,d\,\frac{Wa}{M}$$

$$O = \frac{Wa}{M}\,\frac{\sin\theta\cos(\theta-\Sigma)\,d\theta - \sin(\theta-\Sigma)}{\sin^2\theta}$$

$$\frac{\cos\theta\,d\theta}{} + \frac{\sin(\theta-\Sigma)}{\sin\theta}\,d\,\frac{Wa}{M}$$

$$O = \frac{Wa}{M}\left[\frac{\sin\theta\cos(\theta-\Sigma) - \sin(\theta-\Sigma)\cos\theta}{\sin^2\theta}\right]$$

$$d\theta + \frac{\sin(\theta-\Sigma)}{\sin\theta}\,d\,\frac{Wa}{M}$$

$$O = \frac{Wa}{M}\,\frac{\sin\Sigma}{\sin^2\theta}\,d\theta + \frac{\sin(\theta-\Sigma)}{\sin\theta}\,d\,\frac{Wa}{M}$$

$$d(\theta) = \frac{\sin\theta\sin(\theta-\Sigma)}{\dfrac{Wa}{M}\sin\Sigma}\,d\,\frac{Wa}{M} \qquad [5]$$

If the change in the instrument constant is proportional to the change in temperature, it would appear that the change in reading angle caused by a small change in temperature is proportional to the change in temperature but is also proportional to the function sin θ $(\theta - \Sigma)$. The latter function has a maximum at $\theta = 90 + \frac{\Sigma}{2}$, the reading angle at which the Superdip is at maximum sensitivity.

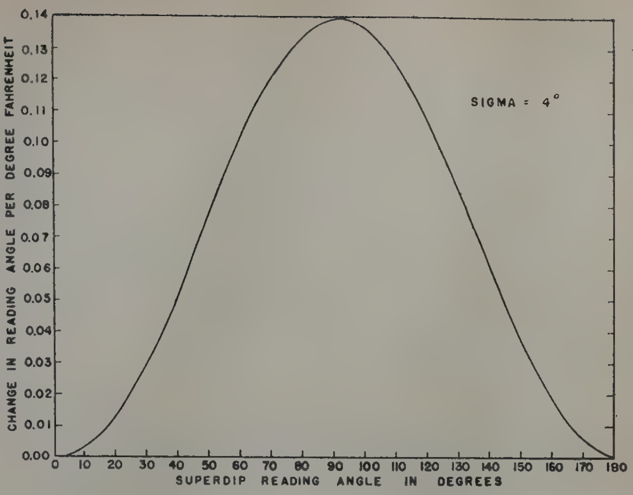

Fig. 5—Temperature correction in Superdip degrees per degree Fahrenheit as a function of the reading angle, Superdip No. 334.

Fig. 5 indicates the fallacy of using the same temperature correction per degree change in temperature for Superdip reading angles over the range of the instrument.

Another approach to temperature correction theory can be made by finding the change in indicated gammas per degree change in temperature.

If eq 5 is divided by eq 3, the apparent change in the vertical field caused by a change in the constant of the Superdip can be expressed thus:

$$\frac{d\theta}{\dfrac{d\theta}{dV}} = \frac{\dfrac{\sin\theta\sin(\theta-\Sigma)}{-\dfrac{Wa}{M}\sin\Sigma}\,d\,\dfrac{Wa}{M}}{\dfrac{\sin^2\theta}{\dfrac{Wa}{M}\sin\Sigma}}$$

$$dV = -\frac{\sin(\theta-\Sigma)}{\sin\theta}\,d\,\frac{Wa}{M}$$

Therefore, the apparent change of the vertical field caused by a change in temperature is proportional to the change in the instrument constant and is also proportional to the function $\frac{\sin(\theta-\Sigma)}{\sin\theta}$. The latter ratio is much more nearly constant in the range of reading angles from $\theta = 20°$ to $\theta = 160°$ than is the function sin θ sin $(\theta - \Sigma)$.

Table III. Partial Table, Calibration Data

$\theta°$	$\frac{\sin(\theta-4)°}{\sin\theta}$	$\frac{Wa}{M}\,\frac{\sin(\theta-4)°}{\sin\theta}$	Station Value	$\theta°$	$\frac{\sin(\theta-4)°}{\sin\theta}$	$\frac{Wa}{M}\,\frac{\sin(\theta-4)°}{\sin\theta}$	Station Value
4	0.000000	0.000000	—55,240	94	1.002442	59,505	4265
5	0.200239	11,890	—43,350	95	1.003667	59,580	4340
24	0.840887	49,920	—5320	124	1.044614	62,010	6770
25	0.847971	50,340	—4900	125	1.046408	62,110	6870
44	0.925330	54,930	—310	144	1.093577	64,910	9670
45	0.927807	55,070	—170	145	1.097187	65,130	9890
64	0.963541	57,200	1960	154	1.140586	67,700	12460
65	0.965036	57,280	2040	155	1.147149	68,100	12860
84	0.990232	58,780	3540	164	1.240835	73,660	18420
85	0.991461	58,855	3615	165	1.257898	74,670	19430
90	0.997564	59,215	3975	174	1.661258	98,610	43370
91	0.998782	59,290	4050	175	1.794874	106,540	51300
92	1.000000	59,360	4120	179	4.994041	296,450	241200

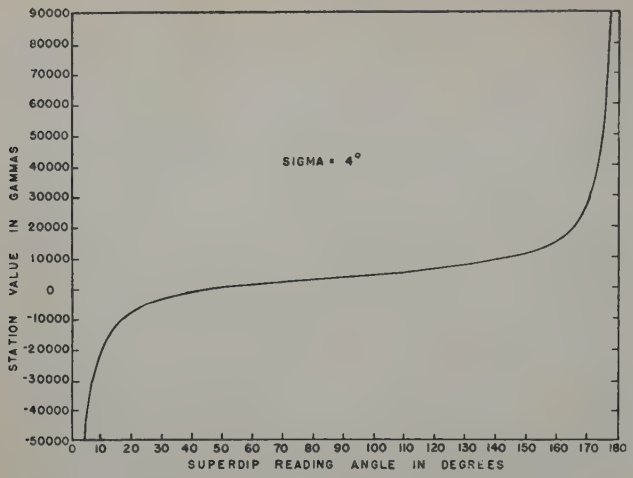

Fig. 6—Temperature correction in gammas per degree Fahrenheit as a function of the reading angle, Superdip No. 334.

Comparison of Figs. 5 and 6 would indicate the advantage of correcting for temperature changes of the Superdip in terms of gammas per degree change in temperature instead of change in reading angle per degree change in temperature. Either method can be used; in any case the actual temperature correction curve would have to be obtained by actual test of a given Superdip at various temperatures in a magnetic field whose time variations are measured carefully.

Reduction of Field Notes

As in magnetometer surveying, the use of base stations is essential in Superdip work. These bases are usually permanent installations in easily accessible locations with station values measured in gammas, as described previously.

In a given day's operation of the Superdip, a base station would be occupied at the beginning of the day, at the end of the day, and at several times during the working period. At all stations, both base and field, the reading angle, the temperature, and the time are recorded.

The reduction of the field notes is made in three steps. 1—The Superdip readings are converted to gammas by use of the conversion curve. 2—The temperature correction is applied to reduce all readings to a common temperature. 3—Correct for diurnal variation of the magnetic field and adjust the level of the results to the base station value. For example, after the first two steps, the indicated station value of the base, first observation, might be 80 γ lower than the true value. The second observation might indicate a value 60 γ too low. Thus 80 γ would be added to the first base observation, 60 γ to the second, and intermediate values to the intervening station observations, treating the diurnal effect as if it were a linear variation.

The final result for each field station is the measured station value, which is used in plotting profiles and isoanomalic contours. Part of an area could be surveyed with a magnetometer and another part of the area surveyed with the Superdip, but since the Superdip data were converted to equivalent magnetometer results, there would be no discontinuity in the mapping.

Accuracy of Station Values

No systematic attempt has been made to check the accuracy of results over the full range of the calibration curve. However, many magnetometer stations have been evaluated by means of the Superdip. In general, the correlation between the two instruments has been very good, especially for the middle portion of the range of the Superdip. The lack of agreement between station values as determined by the Superdip and by the magnetometer has seldom exceeded 15 γ for intermediate station values. A few stations have been checked with the Superdip where either low or high station values necessitated the use of auxiliary magnets with the magnetometer. In these cases, the lack of agreement between the instruments was somewhat greater, but the percent error was not inconsistent with the results at stations of intermediate value.

Conclusions

The Hotchkiss Superdip, oriented normal to the magnetic meridian, can be calibrated and used as a vertical intensity magnetometer of infinite range, so that all magnetic fields encountered may be mapped. Either relative or absolute vertical intensity may be determined.

Careful calibration of the Superdip at least once each field season, and occasional comparison of the field results with magnetometer data can insure an accuracy high enough for ordinary magnetic survey work in iron ore prospecting.

The conventional method of correcting Superdip readings for temperature changes is considered inadequate; therefore it is believed advisable to make temperature corrections in accordance with the theory presented in this paper.

Acknowledgments

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The Metal Mining Industry in Japan

by Robert Y. Grant

This study outlines the history of metal mining in Japan and the characteristics of the industry as they existed from 1925 to 1945. Mining and milling operations are described together with the role of the Japanese Government. A description of post-hostilities conditions and the present state of the industry is included.

MANY of Japan's larger and better known mines date back to the early centuries of the Christian Era. The first stimulus to mining in Japan came through the growth of Buddhism, A. D. 600 to 700, and the attendant need for copper and other metals for temple ornaments and statues. During the Feudal period, 1333 to 1568, mine operations were expanded to meet the demand for ornament metal and for financing of campaigns. After removal of the seat of government from Kyoto to Tokyo, about 1600, the Tokugawa Shogunate placed great emphasis on mining, and many new mines were opened, some of which are still operating.

Toward the end of the Tokugawa period, mining had declined sharply from its earlier peak. The easily worked, high grade portions of the orebodies had been exhausted, and the mining techniques then available could not exploit the deeper, lower grade ores. Similarly, extraction of metal from the sulphide ores presented a major problem.

With the Meiji Restoration in 1868 and the rapid opening of the country to western ideas came a rejuvenation of the mining industry. Mining, milling, and smelting processes were greatly improved, resulting in the extension of development work into hitherto inaccessible portions of orebodies.

The demand for metal during World War I stimulated activity throughout the industry; copper mining, particularly, expanded to a point where Japan ranked second among world producers.* During the depression following the war, mining underwent a severe contraction in activity, recovering slowly during the late 1920's only to undergo a further setback in the worldwide depression which started in 1929. Mineral output declined in 1930, but the beginning of Japanese expansion, 1931 to 1932, brought about an increase in the output of mine products starting between 1932 to 1934. Production continued to rise, the peak output being attained 1939 to 1943 in virtually all commodities. The greatest production of iron ore, chromite, and manganese was not experienced until 1944 when blocked imports, coupled with strong demands for steel, provided the needed incentive.

Through necessity, Japan produced much of her mineral needs from her own mines at high cost during World War II. Taking the 1932 to 1936 period as an average, however, Japanese mines produced only 16 pct of the iron available to industry, 8 pct of the lead, 33 pct of the zinc, and 68 pct of the copper.

Nearly all of Japan's output of gold, silver, copper, lead, and zinc between 1925 and 1945 came from 30 to 49 pct of the mines for which production data are available; for example, 66 of 174 gold-silver mines accounted for 98 pct of the output of this group. Further analysis of mine output indicates that a large part of Japan's metal output was derived from a few large mines; about 10 gold-silver, 15 copper, 3 lead-zinc, 5 manganese, and 2 iron mines account for most of the production. Many of these mines have been operated for 200 to 400 years. With the

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* John J. Collins: *Copper in Japan*, Natural Resources Section Report No. 106.

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This page of Transactions is a continuation from p. 896. The missing pages appeared in Mining Engineering.

exception of the mines in Hokkaido, where settlement took place late, four of the five largest gold-silver mines were opened before 1660 and seven of the eight largest nonferrous metal mines were opened before 1690.

With increasing supply and manpower difficulties, gold mining was drastically curtailed in 1943, largely by cutting off allocation of supplies. About 50 pct of Japan's gold mines were closed and others suspended milling. Many gold mines were stripped of equipment, which was shipped to other mines in Japan or Korea.

Postwar recovery of the mining industry has been impressive, and much of it has been due to substantial loans from the Japanese Government and from funds supplied through various types of subsidy. Withdrawal of government loan support in the latter part of 1949 was more than compensated for by rising metal prices starting in mid-1950.

Between 1925 and 1945 Japanese metal mining operated with an abundant supply of inexpensive labor, a ready market for minerals, government controls, and a fair margin of profit guaranteed by subsidies. With conditions such as these, a study of the characteristics of the Japanese mines and the mining industry during the pre-1945 period will help toward an understanding of the industry in the post-Surrender days. Most of the operational methods discussed below are still in use.

Mining

Ore Grades: During 1925 to 1930, most mines operated on relatively high grade ore, but the better deposits were quickly exhausted, and from an examination of production records it is apparent that most mines in Japan handled low grade ore during the late 1930's and during the period of hostilities. As an example, of total reserves of 25,500,000 metric tons reported by the 38 largest gold-silver mines in 1945, 15,000,000 metric tons averaged 3.0 to 3.9 g of gold per ton, and 5,000,000 metric tons, from 2.0 to 2.9 g per ton. In one of the two largest copper mines, the grade of the milling ore dropped from 0.79 pct copper in 1940 to 0.68 pct in 1945. In the other mine, the crude ore grade dropped from 1.69 pct copper in 1940 to 0.91 pct copper in 1945. These copper mines are underground operations, not open pit.

Except in rare instances, geologic mapping and study in the mines were wholly inadequate to meet the need for more ore. Detailed geologic maps were virtually unknown, and studies of alteration or structure as guides for finding ore were nonexistent.

Extraction, Timbering and Drainage: In spite of low grade ore, mining practice in the metal mining industry was standard except for employment of a large proportion of hand labor. Drills were, in most cases, copies of American equipment, but lighter in weight. The use of wet stopers was almost unknown. A small amount of experimental work on detachable bits was done during the early 1940's, but since bits were not perfected, mines used standard drill steel. Cap and fuse blasting was the universal practice. Because of the lack of control in drilling and the mining of narrow veins, considerable overbreaking was experienced in almost all properties. Smaller producers or operators of mines working irregular or small deposits employed hand methods almost exclusively.

Almost no mechanical loaders were used, not even in tunnels or development headings; hand-loading was the rule, with hoe and pan arrangement instead of the conventional shovel or scoop. The loaders rake the muck into a small two-handled pan with the hoe, lift the loaded pan, and empty it into a car. Timbering followed western practice, but almost all timber framing was done underground, with pine preferred.

Pumping practice was normal. Some Japanese mines are extremely wet, and in a few instances hot springs complicate the drainage problem, but the sharp relief facilitates mine drainage.

Haulage and Hoisting: Hand-tramming was practiced to a great extent, being used at mines where maximum annual output did not exceed 200,000 metric tons. Battery locomotives often were used for main haulage in the medium-sized Japanese mines, but in the few mines of over 300,000 tons per year capacity, trolley locomotives were the rule for main haulage; battery locomotives were employed in collecting. In most instances, the cars, including those used on main haulage, had a capacity of from 1 to 2 tons. In some larger mines 3-ton cars were used.

The largest amount of crude ore handled anywhere was in an iron mine which produced 1,147,000 metric tons in 1944. Most mines produced less than 500,000 tons of crude ore. The maximum output of the ten major producers is given in Table I.

Although in a few mines, skips were provided for main hoisting, most mines hoisted cars.

Mine Safety: Safety practice in the metal mines was not satisfactory by western standards, due as much to indifference on the part of the individual employee as to failure of management to require safe operating practices. The lack of interest in safety is reflected in the lack of information on pre-1945 accident rates available from Japanese Government records.

Mining Costs: Cost data are unsatisfactory because of poor cost accounting methods and because of incomplete information on labor force and on subsidy support. The operations of a few of the base metal mines showed an average mining and exploration cost amounting to 63 pct of the total, with hoisting and haulage accounting for 24 pct; pumping, 4 pct; and maintenance and timbering, 9 pct. Between 1930 and 1945 labor in mining operations accounted for 50 pct of the total cost, minerals about 25 pct, and electric power about 5 pct. The remaining 20 pct was divided among overhead charges and various incidental expenses. Among gold-silver mines the same cost breakdown was the rule.

Ore Dressing

Flotation plants were established at most of the larger nonferrous metal mines, but some produced direct smelting ore shipped after sizing and picking. Almost every mill contained a large hand-picking section. Cyanidation was employed at the gold-silver mines.

The largest flotation plant among the base metal mines had a capacity of 3000 tons per day. Among the cyanide plants the largest mill had a capacity of 2000 tons per day, in a mine which produced 740,211 tons in 1942, the year of largest output. Many of the smaller gold-silver mines employed hand-picking as the only concentrating process. Mill equipment was of American design or, in some cases, of American manufacture.

Efficiency: Although of doubtful accuracy because of poor sampling practice and the lack of appreciation by management of importance of maintaining mill records, the reported average recovery (1930 to 1945) in cyanide plants of 90.36 pct for gold and 75.58 pct for silver is of interest. Metal recoveries in flotation plants were variable, depending upon the type of ore being processed. Copper recovery (1930 to 1945) is reported to have been as high as 92 pct. A low of 65 pct recovery was reported for a complex copper ore containing a large amount of clay. Lead recovery varied between 65 and 90 pct. During World War II, the scarcity of reagents and overloading of mills resulted in a lowering of recovery.

Costs: Costs in Japanese mills did not vary widely from those experienced by American mill operators, with a crushing and grinding cost of about 50 pct of the total, and flotation and hand-picking, 30 to 40 pct. Japanese tailings disposal expenses were high, about 6 to 7 pct of the total. Between 1930 and 1945, labor in Japanese flotation plants averaged about 15 pct of overall costs. Materials cost operators 50 pct, and power 18 to 20 pct of the total. A slightly different pattern was found in the cyanide plants where labor costs amounted to 25 pct, materials about 33 pct, and power about 15 pct of operating costs.

The sale of ores and concentrates to smelters was governed by the same general conditions that existed in the United States and elsewhere. Slightly less attention was given to penalties or other charges than has been the practice in the United States smelters, but this is in accord with the general lack of close control. Also, minor items of cost were of less concern to management.

Labor

Causes of differences between American and Japanese mining are to be found in the social organization, the educational system, and economic conditions that exist in Japan. The Japanese metal mine worker usually is a native of the area in which the mine is located. If a mine is abandoned, the workers remain in the area, turning to full-time farming for support. In the older mines, as many as three or four generations of the same family may have been employed in the mine.

An unsatisfactory ratio between the number of underground and surface workers in Japanese mines always has been reflected in excessive office staffs. By 1945 only from 30 to 40 pct of the total labor force was employed underground. This was not caused entirely by a shift from underground to surface but in part by an increase in one segment of the labor group. A sizable increase in service personnel was required with the introduction of food rationing and the expansion of welfare activities occasioned by the departure of heads of families or wage earners for the military services.

Table I. Maximum Output of Major Producing Mines in Japan, 1925 to 1945

Type of Mine	Year	Maximum Crude Ore Output, Metric Tons
Iron ore	1944	1,147,000
Copper	1942	1,007,000
Lead-zinc[b]	1943	983,000
Copper	1944	893,000
Copper	1942	839,000
Gold	1942	740,000
Copper	1943	696,000
Gold-copper	1942	656,000
Pyrite	1938	609,000
Gold-copper	1942	506,000

[a] Source: company records.
[b] Two adjacent mines under same management.

Most Japanese mines employ many women workers, particularly in surface installations. Women commonly were employed underground for haulage and similar tasks in the 1925 to 1945 period and particularly during World War II. Korean and Chinese laborers began to be employed near the end of the 1930's. By 1944, in the Hitachi copper mine, one of the largest in Japan, 1089 workers of a total labor force of 3154 were Koreans. Prisoners of war were used in many of the larger mines during the latter part of the war.

The wage structure followed the usual Japanese wage pattern. The worker received a basic wage plus a number of bonuses and allowances. Attendance allowance, efficiency allowance, high price allowance, family allowance and underground work allowance are a few. Special allowances were expected for marriage, the birth of a child, and deaths in the family, and at the end of the year. In addition to the wages, the Japanese mine worker received benefits from the company in the form of low prices for many items of food and clothing.

Between 1925 and 1945 surface workers received only 60 to 80 pct of the wage of the underground workers. Women workers in four representative Japanese mines received an average wage only 37 pct of the wage received by the male workers who, in many cases, were doing the same or lighter work.

Unions were organized, but rigidly controlled by the companies; they were largely organizations permitting a more convenient hold over the mine population.

On the other hand the paternalistic pattern of all Japanese industry was found in mining as well. The operator was expected to assume responsibility for support of the families in case of injury to workers. Formal agreements were not normally established, nor were compensation payments standardized although compensation laws existed. Pay deductions for welfare or compensation were rarely made. Under such a system, as mentioned above, compensation to an injured worker often took the form of continued support to the family, or in a case of permanently incapacitating injuries, light work in the office was often provided. When mine operations were suspended for a period, the operators continued to support most of the workers and families.

Government and Mining

Prior to the mid 30's the Japanese mining industry was relatively free from government interference, but as requirements for minerals increased, controls began to be introduced. By 1938, most of the mineral and metal prices, as well as the supply of metals, were controlled. Every phase of the industry was supervised by the Ministry of Commerce and Industry† through the Mining Bureau. Authority for such supervision was present in the Mineral Industry Law, the basic mining law of Japan, as well as in emergency legislation. A number of semi-governmental organizations established during and after 1938 permitted the government even greater control by allowing it to enter directly into operations. It was through these groups also that much financial support was provided to the mining industry.

Control Organizations: The most important of the

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† Now Ministry of International Trade and Industry.

Table II. Mine Production by Company, In Pct of Total Output 1948 Calendar Year[a]

Firm	Copper	Lead	Zinc	Pyrite
Mitsui Mining Co.		41	64	
Mitsubishi Mining Co.	28	36	20	5
Nippon Mining Co.	27			10
Seika Mining Co.	8			5
Nippon Zinc Mining Co.		5	5	
Furukawa Mining Co.	13			3
Dowa Mining Co.	8			34
Matsuo Mining Co.				29
Other	16	18	11	14
Total production,[b] metric tons	26,000	6880	33,640	1,138,100

[a] Source: company records.

[b] Metal content of concentrates except for pyrite which is expressed as concentrate.

semigovernmental organizations in relation to the nonferrous mining industry was the Imperial Mining Development Co. Ltd., organized in April 1939. The Japanese Government held 50 pct of the stock of the company. Imperial was authorized to enter into any activity related to mining, including exploration, mine operation, granting of loans, and even operation of such auxiliary facilities as aerial tramway manufacturing concerns. The Japanese Government was to provide funds where necessary to guarantee a 4 pct profit to shareholders during the first 5 years of operations and 6 pct thereafter.

A second organization that worked closely with the Mining Bureau was the Metal Mining Control Assn., established in 1941, with a membership made up of nonferrous mining companies. It was formed to work out the solution of production problems and was an outgrowth of four control associations, the Copper, Lead-Zinc, Tin, and Mercury Control Associations which had functioned since March 1938. A Metal Distribution Co. which operated in conjunction with the Metal Mine Control Assn., was the distributing agency for nonferrous metals.

Government control in the ferrous field was provided by the Iron and Steel Assn., and by such groups as the Imperial Manganese Co. (later the Imperial Manganese and Chromite Co.) with functions similar to those of the Imperial Mining Development Co. and the Metal Mining Control Assn. Tungsten, molybdenum, pyrite, and gypsum were brought under control in 1943 when the Imperial Manganese and Chromite Co. was redesignated the Ore Distribution Control Co.

Financial Support: An intensive program of monetary support was started in 1938, partly to stimulate production and partly in apparent recognition of the lower grade ore, smaller reserves, and rising costs. The subsidy program also permitted the government to maintain control over domestic price levels by covering operating losses through subsidies rather than through price raises. The effect of the strict control over prices of metals, in spite of increasing costs, is illustrated by small rise in the official copper price between 1938, when it was Y1,050 per ton, and 1945 when it had risen to Y2,060 a ton. This is in marked contrast to the situation after 1945 when the price of copper rose from the 1945 price of Y2,060 to Y181,060 a ton in 1948.

The subsidy program gave direct financial support to the mines covering up to 50 pct of exploration and mill construction costs. In addition, subsidies were provided on gold and other metals. With the yen price of gold at Y3.85 per g, production bonuses from Y0.80 to Y3.30 per g were paid. Similar bonus plans were started in 1943 covering other metals. The amount of money paid to a mine had little or no relation to metal output. In 1943 one marginal property received Y1,279,809 in metal price bonus funds for a production of 624 metric tons of copper in concentrates; during the same year, a second mine received Y1,327,788 while producing 8343 tons.

Funds also were provided to the industry in the form of loans to the mining companies from the semigovernmental organizations, such as the Imperial Mining Development Co. Between 1939 and 1945, during a time that the yen-dollar exchange was about $0.25 to Y1.00, about Y333,000,000 in loans were made to the mining industry; most of this has not been repaid. Large amounts of money were made available to metal mining through the purchase by the Imperial Mining Development Co. of gold properties after suspension of gold mining in 1943. More than Y230,000,000 was paid for the mines although only Y4,000,000 was made available as cash. The remaining funds were deposited in blocked accounts. The blocked funds could be utilized by the mine owners for the payment of taxes and other government obligations or for the support of the operation of lead-zinc or copper mines. This effectively drained the blocked accounts. Many of the mining rights so acquired are still held by the government.

The total amount of financial assistance made available to the nonferrous mining industry through 1939 to 1945 cannot be determined accurately, because of loss of records and confusion in respect to financial transactions. Available data on a sizeable segment of the industry indicate that at least Y800,000,000 were introduced as new capital, and it may be that half again as much was furnished to the industry. The importance of such financing cannot be discounted. Records of one copper mine show that 50 pct of the gross income of the mine was derived from the Government in 1943.

During 1945, mine output dropped sharply, with almost complete suspension of operations after the end of the war. Most of the gold mines failed to reopen, with only 14 of the 40 largest being in operation in 1946 and only 18 in 1947, falling off to 16 in 1948. Nonferrous metal mines, for the most part, resumed operations, with 26 of the 32 largest in operation in 1946; 27 in 1947, and 26 in 1948. Small mines began operations also, but the number was greatly reduced; by December 1948, only 159 copper mines reported production, and 177 in 1949 in contrast with over 300 during 1935 to 1940. Very few of Japan's iron mines were able to reopen because of the low grade of ore reserves.

In spite of slow initial recovery caused by financial difficulties, labor problems, shortages of supplies, and uncertainty within management circles brought about by the threat of deconcentration, by the end of 1949 substantial recovery had been made. Production of most metal was rising and plant expansion was being undertaken. Although some setback was experienced with the removal of price controls and elimination of government loans, rising metal prices and increasing demand for metals of all kinds beginning in the spring of 1950 gave added impetus to the recovery movement.

At present most metal mining companies show a clear profit from their operations, and mines which have not operated since 1944 and 1945 are being reopened. Competition for ore supplies is growing, with some companies constructing new mills or adding to old ones for the primary purpose of treating custom ore.

The worldwide scarcity of metals and strong competition in world markets for ores and concentrates make it probable that Japan's industrial machine will find it necessary to depend in large part upon the domestic mining industry for basic raw material supplies. For this reason, in spite of low grade ore and small deposits, Japan's metal mining industry should enjoy profitable conditions as long as rearmament programs continue to occupy their present important position.

Equipment and Supplies

With a fuel shortage during 1946 and 1947 curtailing industrial recovery, machinery and equipment were difficult to get and expensive. Fuel, lubricants, steel pipe, air and water hose, electric wire, steel plate, and conveyor belts were in short supply.

The 16 pct of all mine equipment deadlined on Jan. 31, 1947, for lack of spare parts and material, was one effect of the shortage.[‡] Other equipment was being run at reduced levels to avoid the frequent breakdowns resulting from capacity usage. Operators reported that at least 70 pct of all equipment was at least 5 years old. Coupled with the shortages was the poor quality of much of the material available; as an example, wire rope was being worn out in from 6 to 9 months.

The shortage of supplies continued through 1947. The nonferrous metallurgical industry reported that the items found to be short in the mining industry in 1946 were still out of stock at the end of 1947.[§]

By 1949 coal production had increased to a point where most industries were receiving reasonably adequate supplies. Equipment and materials became available in ever increasing quantities with the result that long delayed rehabilitation and development work could be undertaken. In addition to the normal replacement of worn machinery, some properties undertook large scale changeovers in mining methods. The use of scrapers was started at several mines; new drilling methods were instituted as new and better drills became available. A completely new underground ore-handling system and crushing plant were installed at the Yanahara pyrite mine, Okayama Prefecture, the second largest in the country. New mills were built and the flowsheets of old mills revised to include heavy media or other newly developed equipment. Although by July 1, 1951, some effect of the Korean affair was noticeable in the availability of equipment, in large measure little difficulty was being experienced by mine operators in purchasing new machinery.

Unions and Labor Supply

Coupled with high prices, shortages, and high costs, labor difficulties contributed to the slow recovery of the mining industry. Metal mine unions have not been as troublesome as coal mine unions, but as might be expected, with the introduction of the new concept of labor unions free from control by management, confusion arose with respect to the rights of management and unions. At first union officials received full-time pay from the mining companies, although they spent all their time on union activities. Management personnel were forced to spend many hours, which could have been spent to greater advantage in supervision operations, negotiating over union complaints, most of which were minor and many of which had to do with activities properly the concern of management. Labor supply is adequate for current needs.

Unions: Most of the workers and unions belong to the All Japan Federation of Metal Mine Workers, organized in May 1947 with 126 member unions and 70,000 members. It had a claimed membership of 83,000 in December 1948. Leftwing influence has been strong but not dominant. Following early efforts to achieve industry-wide action, which failed to materialize, on wages and working conditions, recent union bargaining has been limited to discussions and several work stoppages which rarely spread beyond unions within a single company. Strikes affecting the entire industry have been held, but these also were merely more or less simultaneous action by single mines, in some cases apparently for the sole purpose of saving face by not being left out of the strike action.

Labor Supply and Distribution: At the end of 1950 the Japanese Mining Bureau reported that 86,633 workers of all classes were employed by the metal mining, smelting and refining industry, not including the iron and steel plants. In 1949, 80,498 were employed; of these, 37,100 were classed as surface workers. From this it appears that the same general distribution of labor still is in effect as that found at the end of World War II. Slow improvement has been made, but during the month of September 1947, a large (for Japan) copper mine reported an underground force of 803 (34.6 pct) of a total labor force of 2321, not including staff technicians, while producing 12,000 tons of ore. Elimination of loans has forced the removal or shift of some surplus labor, although this is against Japanese traditional practice. The same copper mine cited above had 2420 employees classed labor as differentiated from staff in 1949, with 1042 underground (43.0 pct), with an average monthly output of 27,700 tons of ore.

Deconcentration

Another delaying factor in the recovery was the threat of reorganization hanging over most of the mining companies. The original policy established for the Supreme Commander for the Allied Powers to follow in dealing with the companies was to require major deconcentration and reorganization. With a clearer understanding of the limitations of

Table III. Nonferrous Metallurgical Plant Capacities by Company, In Pct of Total Capacity, 1948 Calendar Year[a]

Firm	Copper		Lead		Zinc	
	Smelter	Refinery	Smelter	Refinery	Electrolytic	Retort
Mitsui Mining Co.	5.0	7.3	29.0	37.0	51.0	100.0
Mitsubishi Mining Co.	20.6	18.6	34.0	21.0	30.0	
Nippon Mining Co.	30.8	20.8		31.0		
Nippon Soda Mining Co.			28.0	10.0	18.0	
Chugai Mining Co.			9.0	1.0		
Toho Zinc Co.		0.8				1.0
Furukawa Mining Co.	10.7					
Dowa Mining Co.	9.2	8.3				
Seika Mining Co.	19.7	22.8				
Dai Nippon Mining Co.	4.0					
Furukawa Electric Co.		21.4				
Total production, metric tons	120,940	115,560	21,100	42,725	29,620	26,000

[a] Source: company records.

[‡] Michael B. Nicholson: Natural Resources Section Preliminary Study No. 14, Machinery Distribution in the Japanese Mining Industry (July 19, 1947).

[§] Michael B. Nicholson: Materials and Power used in the Japanese Nonferrous Smelting and Refining Industry, Natural Resources Section Preliminary Study No. 32 (March 26, 1949).

Table IV. Subsidies and Loans to Metal Mining Industry[a]

Japanese Fiscal Year[b]	Subsidies				Government Loans		
	Gold Exploration[c]	Copper or Other Exploration[c]	Pyrite Price Adjustment[d]	Copper, Lead, Zinc Price Adjustment[d]	Equipment and Housing[e]	Operating Funds[f]	Counterpart Aid Funds[g]
1946[h]	2,000,000	None	66,038,000	None	19,000,000	345,179,000	None
1947	4,200,000	None	None	801,414,000	374,001,000	796,421,000	None
1948	14,616,000	None	None	2,675,712,000	881,847,000	522,475,000	None
1949	14,440,000	59,337,200	None	None	None	None	308,000,000
1950	ND[j]	151,769,180[k]	None	None	None	None	80,000,000

[a] Source: Gold subsidy and loan data from Mining Bureau, Japanese Government and Mining Industry Association; price adjustment subsidy data from Bank of Japan.
[b] April of named year through March of following year.
[c] One half the cost of exploration (underground).
[d] Difference between consumers price and pool of average cost of three or more mines producing same mineral plus dealer's price and Kodan (Public Distribution Corp.) charge is paid as subsidy to the mining company.
[e] Includes a small amount for nonmetal mines.
[f] Includes funds for mining, milling, smelting, and refining.
[g] Matching yen fund based on amount of dollar support to Japan.
[h] July 1946 to March 1947.
[i] Includes some funds estimated on basis of budgeted items.
[j] Included with copper exploration.
[k] Includes gold subsidy of about Y14,000,000.

the industry, however, came relaxation of the original policy. Control of the mining industry actually had been fairly well balanced. The situation at the end of 1948, shown in Tables II and III, illustrates this point.

When the reorganization of the mining companies was finally directed on August 30, 1949, only Mitsui, Seika (Sumitomo), and Mitsubishi mining companies were affected. Even then the relatively good balance between the companies was recognized, so that these companies were required only to split into two companies, a coal, and a metal mining and metallurgical company. The separation of the coal and metal mining activities of the companies involved is logical because the coal mines are in no sense captive mines—the companies use but a minor part of the coal produced in their own metallurgical plants. In fact, Mitsui controlled about 16 pct, Mitsubishi 11 pct, and Seika about 4 pct of the national coal output for 1948. The break-up ordered should have no detrimental effect on the industry, and in the case of at least one concern improvement in operations has resulted.

Finances

General: With severe postwar inflation, which can be measured by the price of copper, which rose, as stated earlier, from a 1945 price of Y2,060 to Y181,060 a ton by 1948, rehabilitation and reconstruction work was slow in starting. Mine operators were reluctant to expend inflated yen. Reconstruction of several mills was delayed or suspended entirely, pending return to more or less normal conditions. In other instances, in mines formerly having large mills, the capacity of new plants was sharply curtailed; instead of a 6000 ton per month plant, burned in 1945, a copper mine has installed a 2500 ton per month plant. A gold-silver mine formerly having a 3000 ton per day mill capacity constructed a mill of 400 ton capacity.

Operating costs generally rose more rapidly than the controlled metal prices. Some of this difficulty was due to the fact that, while the mines found it necessary to obtain many items of supply through illegal and expensive channels, the cost of production on which metal prices were based was calculated by the pricing agency on the basis of official controlled prices of materials.

To provide funds for the mining industry to use in rebuilding and for meeting operating expenses, and in the absence of sufficient private capital, the Japanese Government took over the task of financing by subsidies and loans. During the same period every effort was made to hold metal prices down through both price and distribution controls.

Subsidies and Loans: Initially the primary needs of the industry were funds to pay operating expenses and meet payrolls. Thus in 1946, Y345,179,000 in loans were provided, largely from the Reconstruction Finance Bank of the Japanese Government, to be used to defray operating costs. A small amount was made available for housing and equipment. The sum of Y66,038,000 was paid to pyrite producers as a price adjustment subsidy to cover production costs, which were higher than the selling price.

By 1947 rehabilitation programs were developing rapidly and with them came the demand for funds in greater amounts. During this year loans of Y374,001,000 were provided for housing construction and equipment purchase and more than Y796,400,000 for operating expenses. Price adjustment subsidies were furnished copper, lead and zinc producers in the amount of Y801,000,000.

Government financing reached a peak in 1948 when more than Y4,000,000,000 in loans and subsidies were paid to mining companies. Action taken by the Supreme Commander for the Allied Powers ended most government loans to the mining industry in the fall of 1949. More money was allocated to exploration subsidies, however. The Nippon Mining Co. was granted loans from United States Aid Counterpart Funds for sulphuric acid plant construction at two smelters to utilize waste gases. The detailed breakdown of loans and subsidies provided to Japanese metal mining from 1946 to 1951 is given in Table IV. It should be remembered that the yen exchange rate has changed several times, so the subsidies and loans are not as formidable as they appear to be at first glance. From a pre-Surrender Y4.00:$1.00, the exchange rate has risen until the rate on April 23, 1949 was Y360.00:$1.00 at which point it has remained through July 1, 1951.

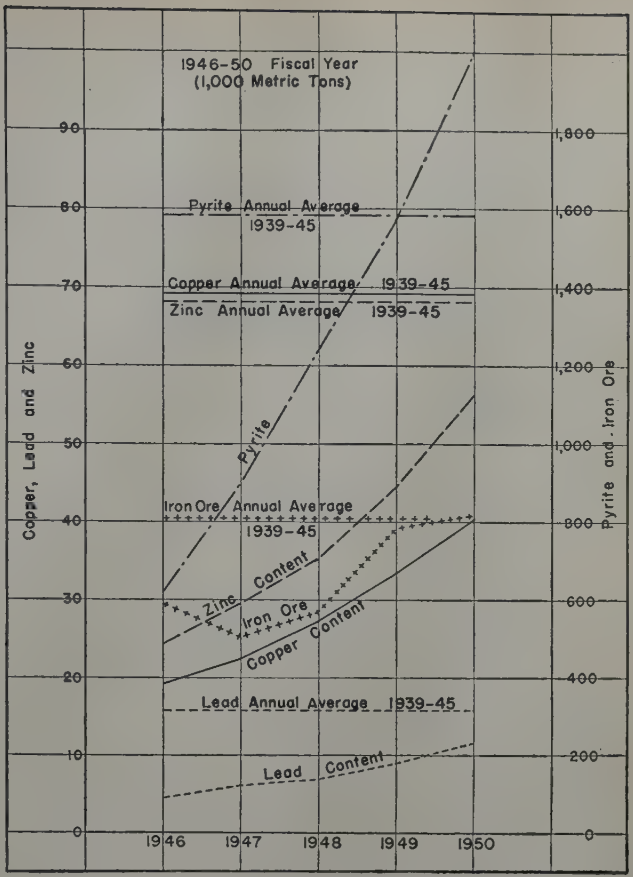
Although metal mining received considerable assistance, the coal mining industry was in a far more favored position than the metal during the postwar period, with subsidies and loans amounting to Y2,100,000,000 in 1946; Y19,300,000,000 in 1947, and Y33,300,000,000 in 1948, as compared to funds received by nonfuel mining industry of Y432,217,000 in 1946; Y1,976,036,000 in 1947, and Y4,064,650,000 in 1948.

It should be noted that Japan's Government organization with respect to mining provides little in the way of the indirect assistance available in the United States. The Japanese Mining Bureau operates under the Resources Agency of the Ministry of International Trade and Industry and is almost exclusively an administrative organization. In 1948 the entire research budget amounted to only Y5,983,000 of the Mining Bureau's budget of Y850,000,000. Completely divorced from the other agencies concerned with underground resources is the Geological Survey under the Agency of Industrial Technology, Ministry of International Trade and Industry. The Survey has begun to provide some assistance to mine operators needing geologic advice or assistance. Peculiarities of the government mining organizations are that short time officials remain in any one position and the fact that the administrative officials do not remain in the same or related fields of activity. On August 1, 1951 the Chief of the Japanese Mining Bureau was transferred, after serving for 26 months, to the position of Chief, International Trade and Sundries Bureau (pulp, rubber goods, toys, etc.). On the same date the Chief, Mine Safety Bureau, was transferred after serving 5 months, to head the Iron and Steel Bureau (iron and steel production).

Controls and Prices: Although price controls were maintained after the end of World War II and price raises were made subject to the approval of the Occupation agencies, the general inflation brought about rapid price increases. The case of copper has been cited. Lead prices rose from Y1,800 a ton in 1945 to Y80,810 in 1948, zinc from Y2400 to Y58,036, pyrite from Y20.16 to Y2000 and iron ore from Y50 to Y1,154. By 1948 however, with increasing industrial production inflation had been brought to a virtual standstill, metal prices remained at the 1948 level through March 1949 and into 1950.

In conjunction with a general move to free the economy from unnecessary restrictions and governmental direction, many commodities which had been under both price and distribution control were released during September to October 1949. Among those released were copper, lead, zinc, and mercury; iron ore and pyrite controls were retained. As a result of the decontrol, the mining industry for the first time since 1938 was operating under relatively free economy.

Metal prices subsequent to decontrol reflected the production and stock situation rather well. Copper prices broke from Y181,060 because of large stocks derived from the smelting of scrap and curtailed spending by the Japanese National Railway, the largest copper consumer. Several large lots of copper were exported at prices well below world market in an effort to free capital tied up in copper. Mercury prices dropped as well, reflecting small demand and relatively large stocks. The demand for zinc remained strong through 1949 and 1950, resulting in a doubling in price. At the end of March 1951 pyrite and iron ore were decontrolled, and the prices of these items started a slow rise.



Source: Ministry of International Trade and Industry

Fig. 1—Mine production, copper, lead, zinc, pyrite, and iron ore.

Table V. Japanese Metal Prices and Price Decontrol Data, Yen Per Metric Tons[a]

Metal or Ore	July Metal Price, Yen Per Metric Ton 1949	1950	1951	Decontrol Data and Price Prevailing
Copper	181,000	170,000	300,000	Oct. 1, '49, @Y181,060[e]
Iron Ore[b]	1,298	1,298	2,600	Mar. 31, '51, @Y1,800
Lead	80,810	83,000	240,000	Sept. 2, '49, @Y80,810
Mercury	32,000[d]	20,000[d]	78,000[d]	Sept. 2, '49, @Y32,000[d]
Pyrite[c]	1,634	2,064	3,230	Mar. 25, '51, @Y2,585
Tin	520,000	570,000	1,500,000	Dec. 4, '48, @Y99,000
Zinc	58,030	116,000	250,000	Sept. 2, '49, @Y58,030

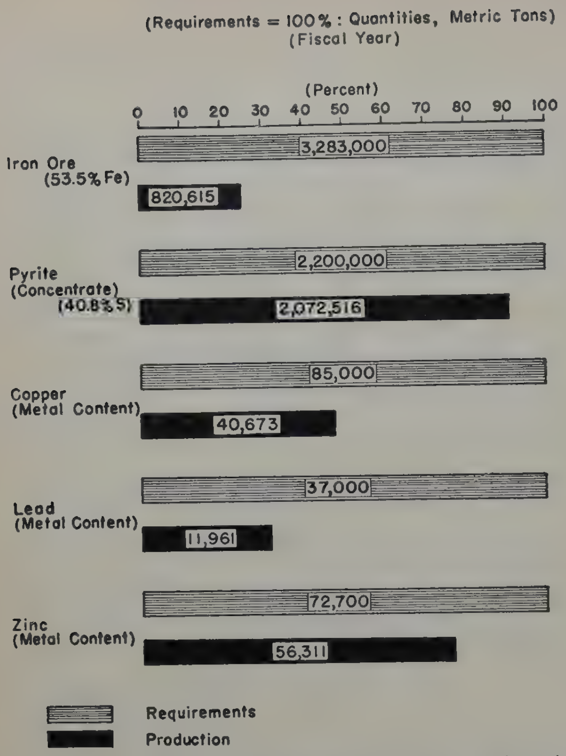
[a] Source: Japan Mining Industry Association.
[b] 50 pct Fe.
[c] 50 pct S.
[d] Price per 34.5 kg.
[e] Consumers price Y102,014.

With the advent of the Korean war, metal stocks largely disappeared and metal prices moved upward rapidly. To illustrate the recent metal price history in Japan, metal prices and decontrol data are supplied in Table V. As commodity prices in general increased less rapidly, the mines are now enjoying a prosperous period.

Production

As measured by the increase in the output of mine products, the recovery of the Japanese metal mining industry since 1945 has been generally satisfactory. Metallurgical plant output, drawing on large scrap stocks and imported raw materials, has been nearly equal to demand and in some cases, prior to the outbreak of the Korean fighting in 1950, exportable sur- such as copper, lead and iron ore, being limited by pluses developed. Mine production of some metals such as copper, lead and iron ore, being limited by small reserves, has been less than demand.

Pyrite mining has made the most rapid recovery because of adequate ore reserves to support output and the demands of the fertilizer industry for more



Source: Production, Ministry of International Trade and Industry
Requirements, Economic Stabilization Board

Fig. 2—Mineral and metal requirements 1951 compared with production 1950.

sulphuric acid. Fertilizer, one of Japan's most critical needs, has received particular attention from all agencies. Contributing industries, such as pyrite mining, have also come under the program. Most of the funds for equipment and housing, listed in Table IV, have been for pyrite mines or mines producing byproduct pyrite.

Mine Output: Between 1945 and 1950, mine production of major ferrous and nonferrous metals rose sharply; however production was far below the average for the 1939 to 1945 period, generally the period of maximum output. Pyrite production is the exception—having gone well above the 1939 to 1945 average. In Fig. 1 is charted the record of mine production 1946 to 1950.

Metallurgical plant output for 1950, when compared with mine production figures, illustrates the role of scrap and imported ores and concentrates in providing Japan with an adequate supply of metal, see Table VI. Since scrap stocks have been in large measure eliminated, Japanese metallurgical plant operators are entering the world market in the search for concentrates.

Table VI. Production of Metal in Concentrates Compared with Refined Metal Output, 1950,[a] Metric Tons[b]

Form	Copper	Lead	Zinc	Iron
Refined metal	89,690	17,211	52,481[c]	1,935,500[e]
Metal in concentrates	40,673	11,961	56,311	434,926
Total from scrap or imported concentrates and ore	49,017	5,250	6,170[d]	1,500,774

[a] Japanese Fiscal Year.
[b] Source: Ministry of International Trade and Industry.
[c] Electrolytic, 35,147; distilled, 17,334.
[d] Surplus.
[e] Pig iron.

Adequacy of Supply: In Fig. 2 a comparison is made between the output of the metal mines for 1950 and the estimated requirements for 1951 to indicate something of the degree to which the needs of the country are being met. Unquestionably pyrite requirements can be supplied, and an increase in zinc production of over 20 pct is possible in view of the strenuous efforts being made to add to plant capacity. With scrap and imported ores, copper needs possibly may be met without imports of refined metal, but lead metal imports unquestionably will be needed as will imports of iron ore.

Japan's metal mining industry undoubtedly will continue to play an important role in her economy. However, the part played by any given segment will be controlled by the ore reserve situation. Japan has reserves sufficient to support production of pyrite for her own needs and perhaps for export, and with added exploration and development, zinc probably can join pyrite. At high cost, most of Japan's copper needs can be satisfied but lead and particularly iron ore are limited in amount. Nonmetallics used in the processing of metal ores, such as limestone and refractory silica and clays except special-purpose high-aluminous clays, are plentiful. Graphite and manganese, although not high grade, are relatively plentiful but not adequate to meet demand. High grade chrome is available in limited amounts, as is refractory chrome. One encouraging feature in the reserve picture is the tremendous improvement in the quantity and quality of geologic exploration work by the mining companies. Little tungsten is available. An important shortage, particularly from the standpoint of the iron and steel industry, is that of suitable coking coal. Although good metallurgical coke can be made from Japanese coals in adequate quantity by proper blending and by use of additives of various types, the use of such coke poses economic and operating problems which are of sufficient magnitude that coking coal will be imported if available, even at high cost.

Occupation Headquarters

After the surrender of Japan, technological assistance to the Japanese metal mining industry was provided by the Mining and Geology Div., Natural Resources Section, General Headquarters, Supreme Commander for the Allied Powers, General MacArthur's Headquarters in Tokyo. The Division organization roughly paralleled that of the U. S. Bureau of Mines, with a Minerals Branch, Metallurgy Branch, Solid Fuels Branch, Petroleum Branch, and a Minerals Economics Branch. In addition to the technical assistance program carried on by the Division, studies were made of the various mineral commodities to provide background information for planning. A series of reports covering the results of the studies have been issued. Through July 1951, 75 reports and supplements and 21 preliminary studies, covering the reserves, mining, and milling of ore petroleum reserves and production, and metallurgy were published. In addition, five special studies have been made covering machinery distribution, administration of the mining industry or mining methods. The reports are available in photostat or microfilm from the Office of Technical Services, U. S. Department of Commerce, Washington 25, D. C. The issuing agency should be cited as Natural Resources Section, General Headquarters, Supreme Commander for the Allied Powers.

Collector Mobility and Bubble Contact

by M. D. Hassialis and C. G. Myers

THE nature of a collector-coated mineral surface has been the subject of some experimentation and much speculation. Various aspects of the problem have been isolated and studied; it is probable, however, that there exist aspects as yet unrecognized, which are needed for a complete description of the system. The recognized elements of the problem are: 1—The nature of the chemical units comprising the film and the nature of the forces holding these chemical units to the mineral surface; 2—The orientation of the chemical units at the surface; and 3—The disposition of the collector units at the mineral surface. This report touches upon all three aspects of the broad problem; it deals, however, with a new and heretofore unrecognized element—the mobility of the collector units in the two dimensions of the solid-liquid interface.

Regardless of the viewpoint adopted as to whether the chemical units comprising the collector film are collector ions or molecules, held either by chemical or physical forces, the writers on flotation theory are in apparent agreement as to the orientation of the chemical units at the mineral surface. Wark and Cox[1] showed that the contact angle at a stable collector-coated surface in a constant chemical environment is independent of the nature of the mineral and of any resurfacing agent; it depends solely upon the collector used. They showed further that the value of the contact angle increases with the number of carbon atoms in a homologous series of compounds. Taggart, Taylor, and Knoll[2] postulated that the collector ion is oriented with the chemically reactive, polar end toward the mineral surface and the nonpolar, hydrocarbon-like end toward the liquid phase. In support thereof they cited the abstraction of p-p^1 dihydroxydiphenylthiourea and of glycol xanthate[3] by galena without a concomitant increase in the contact angle above that of clean galena. It was argued that in these cases the water-repellent layer which normally would be presented to the liquid phase owing to the orientation of the collector ions is overlain by a layer of water-avid groups, the hydroxyl groups. Implicit is the analogous orientation exhibited by a monomolecular layer of an insoluble substance at an air-water interface. It should be pointed out that these facts also are consistent with the concept that the collector units be flat on the surface of the mineral and that the response of the surface to an air bubble is determined by the ratio of the *area covered by water-avid groups* to the *area covered by water-repellent groups;* the areas being weighted according to the relative water-avidity or water-repellency of the groups involved.

The disposition of the chemical units of the collector film at the mineral surface has not received the attention it merits. Taggart[4] concluded from the observed stoichiometry of the ions entering and leaving a surface that the stoichiometry is obeyed in detail at the surface of the mineral. Thus, in the case of xanthate-coated galena, pairs of xanthate ions are attached to surface lead ions. In the case of *chemisorption,* Gaudin and Preller postulated an agreement between the atoms or ions of the substratum (the mineral surface) with those of the superstratum (collector film); in the case of the physical adsorption their position is not clear. In one place they state "the first type (physical adsorption) does not require any agreement between the atoms or ions of the substratum with that of the superstratum" and in another place that "It seems, in fact, as though some slight influence of the substratum is formed even in physical adsorption." The present work shows that in some cases of chemisorption the chemical units of the collector film are in constant surface-bound motion and may not be assigned to any site of the surface, hence there is no permanent association between the chemical units of the film and the atoms or ions of the solid.

The physical property of the collector film that is of paramount importance is its surface energy. Although this property is not susceptible to direct experimental determination, the related contact angle is. Much of the knowledge of the properties of a collector film derives (by deductive and inductive reasoning) from the behavior of an air bubble thereat

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as determined by the contact angle. This knowledge, however, deals with the equilibrium state wherein contact has or has not been made and it does not deal with the transitory states through which the system passes in the attainment of equilibrium. From the practical viewpoint, the latter information may be and probably is the more important knowledge for it is highly improbable that true equilibria are ever established in the complex physicochemical system, which is the operating flotation machine.

To illustrate this, consider a volume of liquid in which a collector-coated particle and an air bubble are immersed. Knowledge of the surface energy of the liquid and of the contact angle is sufficient to calculate the change in the free energy of the system[6] between its initial state and the final state wherein the bubble is attached to the particle. The sign of this change indicates the direction of the trend towards equilibrium and specifies the equilibrium state but yields no information as to the rate and mechanism of the attainment of equilibrium (contact). It is not unusual to find that a system in a state of unstable equilibrium requires an additional push or activation or catalyzing action before it will move towards its equilibrium state. For example, a mixture of two volumes of hydrogen and one of oxygen is highly unstable but will remain as such for an infinitely long time. Yet when activated by an infinitesimal amount of energy in the form of a spark or by a catalyst, it goes to water with explosive violence. This additional push, this activation energy is the key to many successful commercial processes; knowledge thereof derives from a detailed study of mechanism.

The importance of induction time (bubble-contact time) was noted by Sven-Nilsson[7] and by Taggart and Hassialis.[8] The induction time appears to be related to the rate of attainment of the equilibrium contact angle, but the relationship is obscured by the unknown effect of the undetermined and variable compression of the bubble against the solid surface during the induction time. No explanation of this phenomenon has been offered previously. The present authors consider the mechanism of induction time to be as follows: The presence of an air bubble compressed against the solid surface creates a field of force in the intervening region that acts to restrain chemical units of the collector film which attempt to leave the region in the course of their surface-bound motion. Chemical units outside this region that enter in the course of their random motion are similarly restrained. With time this results in a net increase of surface concentration of collector. When this concentration exceeds some critical value, the pressure of the gas in the bubble is sufficient to disjoin the intervening liquid layer and establish bubble contact with the collector-coated surface.

The galena particles used in the experimentation were cut with an alundum disk from large, naturally occurring, single crystals. One face of each particle, referred to as the working face, was a face of the original crystal. The particles were ground with water on successively finer Norton emery polishing papers finishing with No. 4/0. The edges and corners were similarly prepared. The particles were polished with levigated alumina on a glass lap covered with cleaned and denapped Buehler "Selvyt" polishing cloth. Preparation was considered complete when all faces showed no scratches at 150X magnification. Final dimensions of particle No. 1 were 13x9x8.5 mm; of No. 2, 14x14x6.5 mm.

The alumina used in polishing was prepared from hydrated alumina No. C-730 purchased from the Aluminum Ore Co. This was heated at 1000°C for 1 to 2 hr and then levigated in distilled water for 15 to 30 min as desired. The unsettled alumina was siphoned off and concentrated by settling at a pH+7 to a slurry of about 10 pct solids by weight. The slurry was bottled in 250-g batches. All glasswear used was either Pyrex or Nonsol. Contamination by dust or organic matter was watched for and rigidly excluded. The polishing cloths finally adopted were the cotton-base type; no wool-base cloth was obtained which with or without treatment would not contaminate the particles. The polishing cloths were denapped by polishing a rejected particle for about ½ hr using moderate pressures. The cloths were then boiled for several hours in dilute aqueous sodium carbonate, rinsed with distilled water and then allowed to stand for about ½ hr in very dilute aqueous hydrochloric acid. Finally the cloths were rinsed with distilled water and allowed to stand in several changes of distilled water until used.

Particle handling was done with rubber gloves. Prior to use the gloves were dipped in cleaning mixture, rinsed with distilled water, and allowed to stand in distilled water to desorb any remaining traces of acid. The gloves were rinsed again before and after the operator put them on.

All glassware used in the work was Pyrex. The distilled water storage bottles, though not Pyrex, were made of low-solubility glass. Glassware was thoroughly cleaned with cleaning mixture, rinsed with distilled water, soaked with distilled water and finally rinsed prior to use.

Cleaning of the particles was performed on a glass lap covered with cleaned "Selvyt" cloth. The abrasive was fed to the lap from a bottle equipped with a dropper. Initially, copious amounts of abrasive are used; as cleaning proceeds, the volume of abrasive is reduced rapidly until finally only water is used. This being continued until no abrasive is visible on the lap. In this manner surfaces free of slime visible under 500X magnification are readily produced.

The criteria of particle cleanliness established for this work were the absence of a contact angle after an induction time which was longer than that used in the test and the development of a full contact angle to a 1-min induction time after conditioning with 30 mg per liter potassium ethyl xanthate solution. This last test was run at the end of an experiment to show that depression had not taken place during the course of the experiment.

The xanthates used in the experimentations were purified by the methods described in a previous publication.[8]

Mobility of Collector Ions

The results reported herein are the average results of a number of repeat experiments; the experimental procedure being fixed to give reproducibility.

Experiment No. 1—A galena particle was cleaned on all six faces and tested for cleanliness on its working face and one adjacent side by a 1-min contact with an air bubble in distilled water. Failure to develop any cling was taken as evidence of cleanliness. The particle was then conditioned for 1 min in a 30 mg per liter KEtX solution after which it was washed with distilled water and tested in distilled water with an air bubble. Both the working face and adjacent side showed a 60° angle after a 1-min induction time. The working face of the particle was then polished without abrasive, under a continuous stream of water using a pressure which was insufficient to leave a galena streak on the cloth. After 5 min of such cleaning the working face tested 60° after a 1-min induction.

Experiment No. 2—The preceding experiment was repeated with the sole modification that the pressure exerted on the particle when the xanthated working face was being cleaned was sufficient to leave a galena streak on the cloth. The working face and the adjacent side both showed a very slight cling after a 1-min induction. This test was run either as a continuation of Experiment No. 1 or on a freshly prepared particle; the results were the same. Absence of depression was indicated by the fact that particles at the end of Experiment No. 2 developed the normal contact angle after conditioning in xanthate solution.

Experiment No. 3—A galena particle was cleaned until it tested 0° after 1-min induction on all six sides. It was conditioned for 1 min in xanthate solution and all sides tested with an air bubble after a 1-min induction time with the result that all sides showed a 60°, ± 3°, contact angle. The working face was then polished using levigated alumina for perhaps 10 sec, followed by polishing under a flood of water for some 4 to 5 min. The working face and one adjacent side showed 0° after 1-min induction, the remaining faces showed barely detectable clings. Subsequent conditioning of the particle in xanthate solution produced the normal angle on all faces. A blank test was run omitting only the actual abrasion (even though alumina slurry was splashed on the specimen); there was no reduction in the contact angle of any of the faces.

The reduction in contact angle of the sides tested is related to abrasion of the working face, whether this abrasion is produced with or without alumina.

That it is not related to any other part of the test procedure is indicated by Experiment No. 1 and the blank. That it is not depression caused by alumina slimes or by some soluble chemical is indicated by the production of the normal angle upon retreatment with xanthate solution. The reduction in contact angle indicates therefore removal of the collector film that is responsible for the contact angle. Removal of the collector film does not take place by dissolution or chemical destruction. This is shown by Experiment No. 1, by the blank, and by an auxiliary experiment in which a xanthated galena particle was washed with some 40 liters of distilled water over a 3-hr period without any noticeable reduction in contact angle. This auxiliary test also shows that the film is not removed by diffusion into the body of the solid. It follows that the only way in which the collector film could have left the surface is via the galena streak laid down on the polishing cloth. Abrasion of the working face produces a large number of extremely small galena particles having an aggregate surface area, which is large compared to the surface area of the particle. For the collector film to leave the specimen with these minute particles, it is prerequisite that the chemical units of the film must move from all other sides of the specimen to the side being abraded and thence to the particles produced by abrasion. To test this point the following experiments were performed.

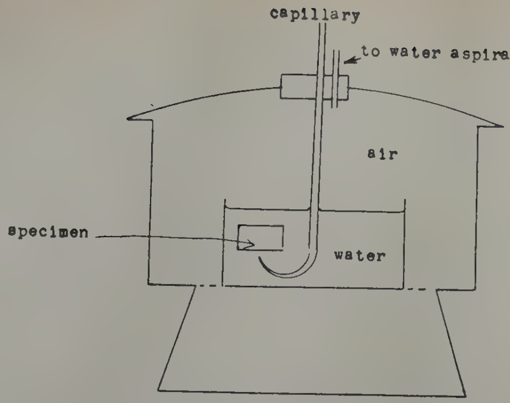

Fig. 1—Equipment for multiple bubble-contact experiments.

Experiment No. 4—Galena particle No. 2 was cleaned until it showed a 0° contact angle after a 1-min induction period on all sides and a 0° contact angle after a 45-min induction time on its working face. It was then conditioned for 1 min in a 30 mg per liter ethyl xanthate solution, after which it showed a 60° contact for a 1-min induction time on its working face. Galena particle No. 1 was cleaned to a 0° angle on its working face for a 1-hr induction period, and a 0° angle after a 1-min induction time on all other faces. Both particles were transferred to a cell filled with distilled water and so arranged therein that their working faces were vertically disposed, parallel to each other and separated by the smallest possible distance just short of actual contact. The system was covered and left untouched for 20 hr, 12 min after which time the particles were removed and tested. Particle No. 2 showed a 60° angle after a 1-min induction time on its working face; particle No. 1 showed a 0° angle after the same induction time on its working face. Particle No. 1 was then conditioned in xanthate solution and developed the normal xanthate angle.

Experiment No. 5—The preceding experiment was repeated with particles of like cleanliness. When the xanthated and unxanthated particles were transferred to the water-filled cell they were arranged so that one particle rested upon the other with their working faces in contact. After 20 min of such contact the particles were removed and their working faces tested. The xanthated particle showed a 60° angle and the unxanthated particle a medium cling, both after a 1-min induction time. They were then returned to the cell and arranged as before. After a total contact time of 17 hr, 13 min the particles were again removed and the working faces tested. The xanthated particle showed a 51° angle and the unxanthated particle a 42° angle, both to a 1-min induction time.

The first of these tests shows that collector film is unable to migrate across a water gap from a coated to an uncoated particle. This might have been deduced from the auxiliary test cited above. These statements should not be interpreted as denial of partition of the collector film between the contiguous liquid and solid phases, but that the partition equilibrium is so far in the direction of the solid as to make negligible the effects of the collector partitioned in the liquid. The second test shows that when physical contact is made between specimens, even though this contact probably consists of a limited number of point contacts, collector film migrates from the coated particle over the solid-to-solid bridges to the unfilmed particle. There appears to be no other way in which this could have happened. Once again, mo-

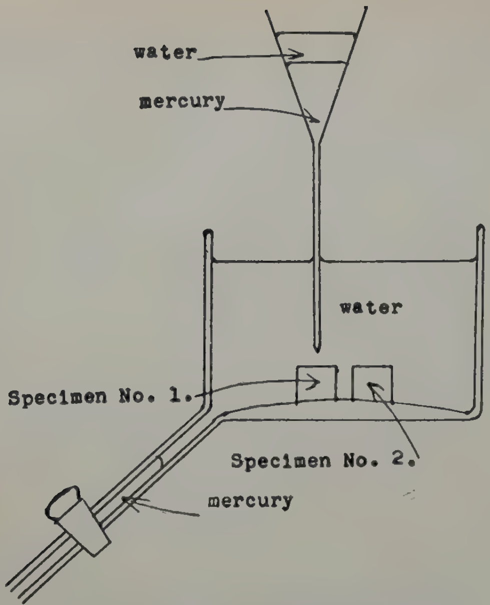

Fig. 2—Equipment for mercury-droplet experiment.

bility of the chemical entities of the collector film in the solid-liquid interface must be presupposed to explain the facts or the facts may be used to deduce the mobility.

Another interesting aspect of Experiment No. 5 is that the amount of xanthate film transferred from particle No. 2 to particle No. 1 produces an increment in contact angle of the latter particle of 42° and a decrement in contact angle of the former particle of only 9°. Even if allowance is made for the difference in surface areas (756 sq mm for No. 2 and 608 sq mm for No. 1) of the two particles, it follows that equal increment or decrement of collector to or from a surface will not produce equal increments or decrements in contact angle. This means that the relation between contact angle and surface coverage or concentration is not linear. It can be further deduced that the rate of change of contact angle with surface concentration is large for small concentrations, decreases with increasing concentration and finally vanishes after some characteristic concentration is exceeded.

The mobility of molecules in a gas-liquid interface[9] and of molecules and atoms in a gas-solid interface[10–13] has been established by many independent experiments. It is curious that no experimental evidence has been advanced in support of similar mobility in the solid-liquid interface. The following group of experiments was designed to determine whether the chemical units of the collector film have the ability to move into and in the gas-liquid interface.

Experiment No. 6—Particle No. 2 was cleaned until it tested 0° after a 1-min induction time. It was then conditioned for 1 min in a 30 mg per liter potassium ethyl xanthate solution, then washed and transferred to a cell filled with distilled water. Particle No. 1 was cleaned until it tested 0° on its working face after a 26-min induction time. It was then placed in the water-filled cell next to but not touching particle No. 2. Both particles were arranged with their working faces horizontal and facing upward. A captive bubble was compressed against the working face of the xanthated particle for 1 min, then removed and transferred to the working face of the unxanthated particle against which it was compressed and maintained for 1 min, finally the bubble was removed and liberated. This cycle was repeated with a fresh captive bubble 150 times. The contact angles for a 1-min induction time were determined for the xanthated and unxanthated particles; at the beginning of the experiment the values were 60° and 0° respectively, at the end of the experiment the respective values were unchanged. By measuring the diameter of the gas-solid contact area and the maximum diameter of the bubble as viewed in the ground glass of the camera of the normal setup, it was possible to estimate the total gas-liquid area and the total gas-solid area; these were about five and two and one-half times the surface area of particle No. 2. At the end of the experiment the unxanthated particle was tested for depression as in previous experiments and none was found.

Experiment No. 7—Particle No. 1 was cleaned until its working face tested 0° after a 5-min induction time. It was transferred to a cell filled with distilled water and there placed with its working face down on a glass tripod. The cell was placed in a vacuum dissicator and arranged as shown in Fig. 1. When the aspirator was turned on, air bubbles released from the tip of the capillary tube rose and collided with the working face, after rebound from this initial collision they proceeded by a succession of bounces to a vertical side of the particle and then escaped to the free water surface where they broke. The bubble rate was determined to be approximately 200 bubbles per min. The projected area of 30 bubbles was determined approximately as 6.1 cm^2. The experiment was run for 3 hr, 7 min, at the end of which time the working face of the specimen was tested and found to have a 0° angle after a 1-min induction time. Depression was tested for and found to be absent.

Experiment No. 8—Particle No. 1 was cleaned to a 0° angle after 1 min in a 30 mg per liter xanthate solution, after which treatment the contact angle was determined to be 60° for a 1-min induction time. Experiment No. 7 was then repeated with this particle using a 60 bubble per min rate, the time of the run being 19¾ hr. Bubble behavior was of three distinct types: 1—Travel across the working face by a series of bounces of progressively decreasing amplitude; 2—rolling along the surface without any visible bouncing, this being the predominant behavior; and 3—rolling, then sticking to the surface until their volume had been increased sufficiently by coalescence, owing to collision with subsequent rolling and bouncing bubbles, to permit dislodgement and escape. It was estimated that the projected area of 60 bubbles equalled the area of the specimen. At the end of the run the working face tested 60° after a 1-min induction time.

It may be concluded from Experiments 6 and 8 that any motion of the entities of the collector film into the gas-liquid interface or partitioning therewith is so small as to be undetected either by a decrement in contact angle of the source particle or by an increment in the contact angle of the unxanthated particle. Experiment 7 is really a blank run, proving that contamination introduced by the air is insufficient to contaminate a clean particle. Without it, interpretation of Experiment 8 would be obscured by the possibility that collector film lost by partitioning in the gas-liquid interface was replaced by air contamination sufficient to maintain the contact angle. The results of Experiment 7 are consistent with the observation that a freshly swept gas-liquid interface takes several minutes exposure to laboratory air before contamination is detectable. It should be noted that the total projected area of the colliding bubbles in Experiment 8 was 1200 times the surface area of the particle. This large gas-liquid area was provided purposely to compensate for a possible difference in ability to transfer under the dynamic conditions of this experiment as compared to the relatively static conditions of Experiment 6.

Mobility of the chemical units of the film in the gas-solid interface was not investigated; mobility across a bridge formed of two dissimilar solids was approached by the following controlled experiment.

Experiment No. 9—Both galena particles were cleaned until they tested 0° after a 1-min induction time on all of their sides. They were then conditioned for 1 min in 30 mg per liter xanthate solution and tested to insure that the normal xanthate angle had developed. Both particles were transferred to the apparatus shown in Fig. 2 and arranged with their working faces up but with neither face directly under the fine capillary tip of the funnel. Chemically cleaned and redistilled mercury,[14] which had been stored under distilled water, was poured into the funnel. The mercury droplets effluxing from the tip of the funnel fell through the water in the cell and collected in the leg. After several minutes the cell was moved relative to the independently supported funnel so that the mercury droplets impinged on the working surface of particle No. 1. The mercury previously collected in the leg was drawn off by means of the water-lubricated stopcock. The rate of dropping was 240 droplets per min. Initially no rebound of the droplets from the working face was observed; they rolled along a relatively narrow path to an edge and then dropped to the bottom of the cell where they collected in the annular depression shown in Fig. 2 and finally in the down leg. Contact between the droplets and the underside of the particles was prevented by seating the particles on the central raised portion of the cell floor. The level of the mercury in the leg was never allowed to rise to the level of the cell bottom, this was achieved by periodic draining of the mercury into a water-filled flask. After 49 min of running time, the cell was shifted so that no mercury dropped on either particle. The contact angles of the two particles were determined in situ and found to be a slight cling for No. 1 and 60° for No. 2, both after a 1-min induction time. The cell was then shifted to bring particle No. 1 under the dropping mercury and the run continued. After some 15 min elapsed time, it was observed that the droplets were rebounding vigorously after their initial collision and were proceeding to the discharge point by a succession of rebounds of decreasing amplitude. When 24 min had elapsed, the experiment was discontinued. The working face and its opposite side of particle No. 1 both tested 0° after a 1-min induction time. Depression was tested for and found absent. The working face of particle No. 2 showed 60° after a 1-min induction time. The mercury which had been collected tested 49° after a 1-min induction time whereas initially it had shown only a light cling for the same induction period. When the collected mercury (260 gm) is poured from one flask into another, always under water, the freshly created surface shows only a strong cling after a 1-min induction time and 46° after a 5-min induction time. The surface of the mercury in the leg during the course of the experiment appeared gray and powdery. Fresh mercury was poured into a cell filled with a 30 mg per liter xanthate solution; after about 1 min most of the xanthate solution was removed, and the mercury surface showed a 61° angle after a 1-min induction time and 70 to 75° for the same induction time 15 min later.

Mobility of the collector film from the xanthated galena particle to the mercury is shown by both the decrease in contact angle of the galena particle and the increase in contact angle of the mercury. That depression is not responsible for the decrease in angle of the xanthated particle is indicated by the maintenance of its contact angle by particle No. 2 and by the ability of No. 1 to regain the normal xanthate angle after retreatment with xanthate solution. The behavior of the droplets at the surface of the particles indicates that in the early part of the experiment mercury-solid contact was made; whether the interface is mercury-oriented xanthate or mercury-galena is not known. It is probably the former, otherwise the bouncing noted in the latter part of the experiment and which indicates the absence of a force restraining droplets to the surface should not have taken place when the surface was freed of xanthate film.

Although mercury has many metallic properties, the preceding experiment does not establish migration across a bridge made of two dissimilar solids. A single preliminary test of the type performed in Experiment No. 5 was made using a copper block cleaned to a 0° angle after a 1-min induction time and a xanthated galena particle. After 15-min contact time the copper block tested a very strong cling (just barely immeasurable) and the xanthated particle 58° both for a 1-min induction time.

Up to this point, reference has been made to the mobility of the chemical units of the collector film. It is pertinent to ask: Are these chemical units xanthate ions or lead xanthate molecules? No definite answer can be given to this question based upon the facts presented. The gray powdery appearance of the surface of the mercury collected in Experiment 9 indicates the existence thereat of an insoluble solid phase, which may be either mercury or lead xanthate. Only in the latter case would argument be advanced to support the thesis that lead xanthate migrated. In the former case, either ion-migration followed by reaction with mercury or lead-xanthate migration followed by reaction with mercury could have occurred.

The time required for the ions or molecules to transfer in the various experiments is unexpectedly long. If it be assumed that the ionic or molecular velocities are comparable in magnitude to those obtaining in liquids or gases, then a fraction of a minute should have sufficed in most cases. If it is assumed that there are bound to the interface other ions or molecules (hydrogen ions, hydroxyl ions, water molecules) and that these also possess mobility in the interface, then the migration of the xanthate ions or molecules is opposed by the counter migration of these other ions or molecules. The resulting collisions, with the consequent shortening of the path between collisions and the changes in direction of motion, will reduce the overall velocity of migration to such an order of magnitude as is associated with diffusional processes; self-diffusion is, of course, always possible. Experiments reported elsewhere[15] indicate that the rate of transfer is not inversely proportional to the square root of the mass of the

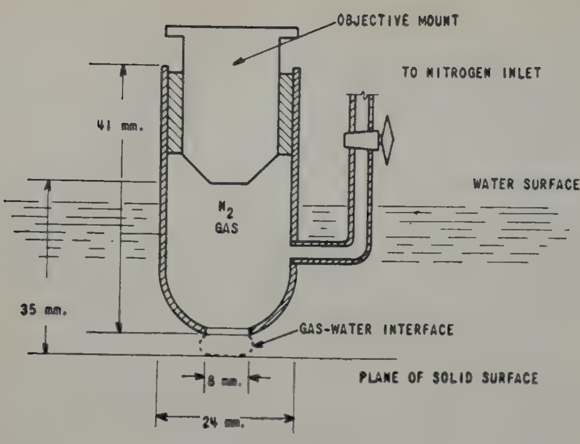

Fig. 3—Bubble holder.

migrating ion as might be expected from kinetic theory.[16] Thus the ratio of the rates of transfer in comparable experiments for ethyl and hexyl xanthate should be inversely proportional to the square root of the ratio of the masses, that is, proportional to 1.21. The experimental ratio is much greater. The migration velocity also possesses an unusually high temperature coefficient; thus a decrease of 25°C from room temperature to 0°C is sufficient to slow the migration of ethyl xanthate ions on galena to a barely detectable rate.

It is too early yet to predict how this new phenomenon will be used in flotation practice; it can be predicted on general principles, however, that insofar as it puts in the hands of the operator another control of the flotation process it is bound to be useful. To illustrate possible utility consider the following example, realizing that it is purely speculative and without any experimental justification: It is desired to separate the molybdenite content of a copper concentrate. Assume that the collector used in the production of the concentrate exhibited mobility or that the temperature of the concentrate is raised to enhance mobility. A small amount of a powder of very high specific surface is mixed with the concentrate and intimate contact produced by stirring. The collector ions migrate from the concentrate to the surface of the powder. The powder is then separated by simple classification or floated off with molybdenite after the addition of neutral oil as in the normal procedure.

Induction-time Phenomenon

It has been shown[8] that for a fixed time of conditioning in a xanthate solution the induction time required to develop the full 60° angle increases as the concentration of the xanthate solution decreases. It was also shown that if, after development of the full angle, the bubble were withdrawn and then immediately returned to and compressed against the same part of the mineral surface, approximately the same induction time was required to redevelop the full angle.

The pressure in excess of the hydrostatic pressure required to maintain an element of area of the bubble in equilibrium is $T\left(\frac{1}{R_1}+\frac{1}{R_2}\right)$,[17] where T is the surface tension and R_1, R_2 are the principal radii of curvature. Since the gas pressure inside a bubble must be uniform, the only way in which elemental areas at different hydrostatic levels can remain in simultaneous equilibrium is for the areas to adjust their curvature, that is, by varying R_1 and R_2. An uncompressed captive bubble is in equilibrium under the combined actions of gas pressure, hydrostatic pressure and curvature. Upon compression the bottom of the bubble is flattened out, that is, R_1 and R_2 are increased (in general by a multiple of the original value). It follows that $T\left(\frac{1}{R_1}+\frac{1}{R_2}\right)$ is considerably decreased, the gas pressure inside the bubble is too large for this area of the bubble. What maintains this area in equilibrium is the reaction of the intervening liquid layer backed by the solid. Since the gas pressure is in excess of the hydrostatic pressure at this point, it follows that any liquid intervening between the bubble and the surface should be displaced. It is not displaced when the solid surface is clean, hence the liquid is held against the solid surface by a force which in combination with the hydrostatic pressure exceeds the gas pressure. It also follows that when the intervening liquid is displaced (as in the case of a collector-coated surface) this force in combination with the hydrostatic pressure is less than the bubble pressure. Since under identical circumstances the gas pressure within a small bubble is larger than the pressure in a large bubble, easier displacement of the intervening liquid layer by a small bubble might be expected. This accounts for the often observed fact that it is easier to obtain contact with a smaller captive bubble.

The induction time cannot be explained by appealing to the viscous resistance of the liquid which would oppose flow of liquid out of the intervening zone. Viscous resistance is proportional to velocity, hence it vanishes at vanishing velocities. Simple consideration of the velocity with which the infinitesimal volume of liquid in the intervening zone must move to require an induction time of say 10 min, renders viscous resistance an improbable cause. The same consideration convinces one that when the bubble exceeds the sum of the hydrostatic and adsorption pressures, the intervening film should move out of the compression zone quite rapidly.

The key to the problem appears to be the change with time in the magnitude of the force holding the intervening liquid layer. This force is manifested in many other phenomena normally classified as adsorption phenomena.[18] It decreases in magnitude as the solid surface becomes coated with a liquid-repellent film.[19] If, therefore, water repellency of the surface can change with time, a probable mechanism is found. The first part of this paper presented evidence proving the mobility of collector ions in the solid-liquid interface. Collector ions located on that part of the solid surface against which the bubble is compressed and in closest contact with the bubble owing to solid surface irregularities are acted on by a force of attraction directed toward and owing to the solid (which force constrains these ions to move in the surface) and a force of attraction[20] directed toward and owing to the gas molecules. These forces have no component in the direction of motion of the ion until the ion, in the course of its random motion, reaches that part of the surface above which the bubble wall is not parallel thereto. At this point the attractive force of the gas molecules has a surface-parallel component directed towards the center of the compression zone. This component acts to restrain ions from moving out of this zone. Ions outside of the compression zone, in the course of their random motion, will enter the compression zone. The net result is in an accumulation of collector ions with time in the compression zone. As the number of these ions per unit area increases, the adsorption force acting on the intervening liquid decreases. When some critical surface concentration of ions is attained, the adsorption force will have been sufficiently decreased that its resultant with the hydrostatic pressure is less than the bubble pressure, and the intervening liquid layer is displaced, that is, gas-solid contact is established.

When the gas-solid contact is broken by withdrawal of the bubble, the collector ions move out of the area of high concentration, and conditions for establishment of contact are destroyed. It was previously noted that the rate of transfer of collector ions is apparently slow; here the evidence indicates that the movement out of the area of concentration is rapid because the usual time elapsed between withdrawal and recompression of the bubble is of the order of 1 to 3 sec. Two factors should be considered in this connection. The surface-concentration gradient existing between the area of high collector-ion concentration and the rest of the surface is high, and change of concentration should therefore be correspondingly high when the restraining effect of the bubble is removed. Even though the change in this concentration in 1 to 3 sec is not great, the time required to reattain the critical concentration when the action of the bubble is brought to bear may be long for in diffusional processes the equilibrium concentration is reached in a manner asymptotic to a line parallel to the time axis. It is not unusual to find that 95 pct of the change in a diffusion process takes place in the first minute, while the remaining 5 pct requires days. The combined action of these two factors may suffice to explain the apparent contradiction noted above.

Another factor bearing on this point is the factor of orientation. As previously noted, most investigators tacitly assume orientation of the collector ions; it was also noted that this condition is not necessary though it may be sufficient. The assumed orientation is open to question on two counts; first, that the potential energy of an isolated xanthate ion on the surface of galena is a minimum when it is lying flat on the surface,[21] second, that there exists within the xanthate ion a second point of attraction for the surface, the oxygen of the ether bond. The flat position appears more probable for an isolated xanthate ion. When the ion is not isolated it may be reoriented to the position normally postulated under the action of either a lateral force of attraction between xanthate ions or a lateral pressure owing to momentum transferred by colliding xanthate ions. Such reorientation would have a marked effect upon the magnitude of the adsorption force acting on the layer of water intervening between solid surface and bubble because it increases the distance over which the force of attraction, exerted by the galena surface upon the water molecules of the intervening layer, would act.

To obtain a clearer understanding of the induction-time phenomenon and of the mechanics of bubble contact, an experiment was devised permitting microscopic examination of the solid surface immediately below the compressed bubble.

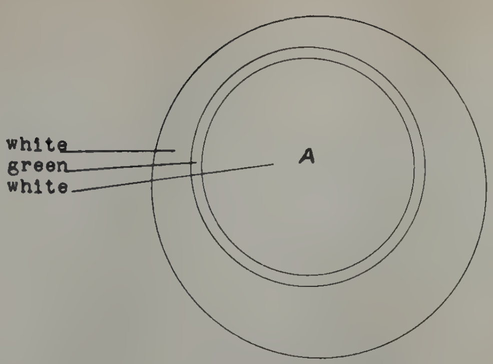

Fig. 4—Initial interference rings.

The microscope used was a Leitz, equipped with a Beck vertical illuminator, a 40-mm objective and a 20x Periplan eyepiece. Total magnification was 64x, working distance 35 mm. A small glass cup, provided with a side arm and opening was attached to the objective as shown in Fig. 3. The cup was cleaned by racking the microscope tube downward until the lower edge of the cup was about 30 mm below the free surface of cleaning solution contained in a cell placed on the stage. As the stopcock was closed during this operation, the cleaning-solution level rose within the cup to a height of only 15 mm above the bottom. Cleaning solution was removed by rinsing repeatedly with distilled water using the same procedure.

Nitrogen cleaned by bubbling through sulphuric acid followed by water is admitted to the cup immersed in a cell filled with distilled water. The gas is turned off and its pressure in the cup adjusted by bleeding so that the distance from the bottom of the bubble formed at the lower edge of the cup to the front lens of the objective exceeded the working distance of the objective by 1 to 3 mm. A clean galena particle placed in the cell was brought into position under the bubble and the stage racked upwards until bubble compression was effected and the surface of the galena came into focus. Within a matter of seconds a narrow green ring appeared in the field of view as shown in Fig. 4. This was followed by a complex sequence of changes within the next minute or two. The original green ring changed color and broadened, simultaneously concentric color rings appeared inside and outside of the original ring. The color cycle of the original green ring went from green to its first red, the first interior concentric ring appeared as green. It then followed a color cycle similar to the initial ring ending with red. While the first interior ring was changing from green to its first red, a second interior ring appeared within the first as green. This underwent a similar color cycle ending with bright blue. While the second interior ring was changing to its first red, a third interior ring appeared as green; it went through a similar color cycle ending with green. While the third interior ring changed to its first red, a fourth interior ring appeared as green, and an exterior (to the initial green) ring appeared as green. The appearance of the interference pattern at this time is shown in Fig. 5. The interference pattern continued its development adding three more exterior rings and four interior rings. At the same time irregular blue gray patches developed in the broad yellow ring, which was the final transform of the single initial green ring. The final appearance is shown in Fig. 6. When

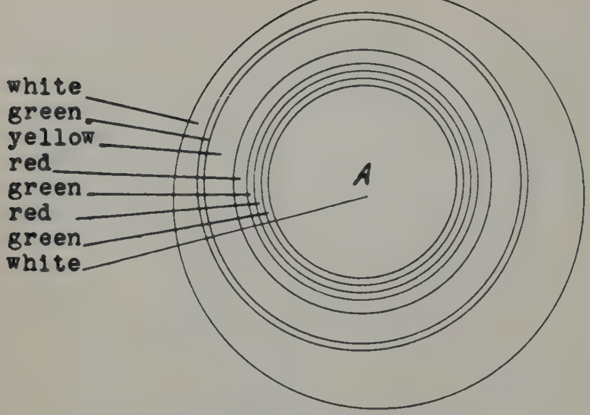

Fig. 5—Subsequent interference rings.

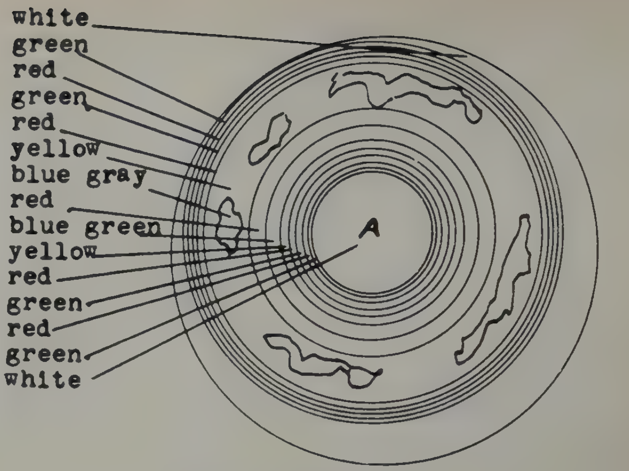

Fig. 6—Island appearance.

monochromatic green light is used, the rings form a succession of alternately black and green rings. The central portion *A* becomes black and fainter, narrower interior rings become visible. Further compression of the bubble had little effect on the interior rings or on the inner boundary of the yellow ring; the outer boundary and the exterior rings were displaced and could be made to vanish from the field of view. Decompression caused the outer boundary of the yellow ring and the exterior rings to reappear and move toward *A;* the interior rings remaining stationary until the outer boundary of the yellow ring and the exterior rings closed in upon them, then all moved in toward A.

It was most difficult to keep the galena particle clean in this experimental procedure. In general, after the pattern shown in Fig. 6 had persisted for some 5 to 10 min, an irregular white spot suddenly appeared within the yellow band and enlarged very rapidly in all directions without regard for the contour of the yellow band, leaving the field free of color and showing the characteristic appearance of galena. This white spot had disposed about it extremely narrow interference bands of similar contour. With the clearing of the field by the enlarging white spot, contact always developed.

It was virtually impossible to obtain the sequence of events recited above using galena conditioned in 30 mg per liter xanthate solution. The operator was fortunate if he could bring the galena surface into focus in time to see the flash of the enlarging white spot. With galena conditioned in very weak xanthate solutions, a portion of the sequence of events was observed.

The circumstances of this experiment are a modification of those requisite for the observation of Newton's interference rings. Two observations are outstanding, the white central spot and the spacing of the bands. In the normal Newtonian pattern the central spot is dark because a relative phase change of π occurs between the rays reflected from the two interfaces.[22] Here both reflections are of the rare-to-dense type since water has a refractive index intermediate to that of air and galena, hence no phase change takes place. In the normal Newtonian pattern, the radii of the rings increases as the root of n or as the root of $(n+½)$ as n assumes successive integral values. In the pattern observed in these experiments, this orderly progression of radii is absent, in fact, the fundamental role is played by the broad yellow ring (the original green ring). There is an obvious difference in geometry of the top interface between the Newton experiment and the present one. In the former, the interface is a part of a sphere, in the latter, it is unknown. Attempts to deduce the geometry from the spacing and width of the rings have led to inconsistencies and must await the development of additional information, particularly since polarization was observed in the image. What these observations demonstrate is that the formation of the gas-liquid interface is rapid when it does occur. This supports the mechanism set forth above.

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Progeny in Comminution

by A. M. Gaudin, H. R. Spedden, and Douglas F. Kaufman

MANY studies of comminution have been made to ascertain the size distribution of the product and to evaluate the work of comminution in the light of the size distributions of the feed and product. Up to now, these studies have been essentially statistical in character, that is, a certain lot of feed was subjected to comminution in some specified way, and the aggregate product was fractionated into sizes, thereby losing all knowledge of individual relationship of feed to product pieces.

Radioactive tracers offer a means to do something in this respect which could not be done before, namely, to follow the rupturing of some particular piece in its normal environment of other pieces. That is, it permits going beyond the usual statistical limitations of size distribution studies to what may be termed a personalized or individualized study. The purpose of this paper is to present some preliminary experiments conducted with this tool.

The method employed was to mark radioactively some constituent of a feed. It is possible, of course, to consider the preparation of two lots of material of which one is radioactive and the other is not, and to blend the two ahead of the comminuting step; but to do so is open to the objection that the two preparations may not be identical. Therefore a technique has been chosen that removes this objection by merely taking out a size fraction of a comminution feed, rendering that fraction radioactive by exposure to a neutron flux, and then by returning it to the remainder of the charge for the comminution experiment.

Table I. Size Distribution of Offspring Albite Particles Originally 28/35 Mesh and in Admixture with Other Sizes After Grinding 2 min in a Steel Ball Mill

Size Fraction of Product, Mesh	Specific Activity Corrected for Background, cpm/gm (A)[a]	Weight, g (W)	Distribution in Product, Pct[b] (P)	Cumulative Distribution, Pct (ΣP)
+28	0	56.0	0	100.1
28/35	62.6	54.0	24.8	75.3
35/48	62.8	59.4	27.7	47.6
48/65	41.1	53.0	16.2	31.4
65/100	29.6	45.7	10.2	21.2
100/150	23.7	37.0	6.6	14.6
150/200	23.3	25.1	4.4	10.2
200/270	20.1	19.0	2.9	7.3
270/400	17.8	21.2	2.9	4.4
—400	22.9	25.2	4.4	—
			100.1	

[a] These activity determinations were made in rapid succession in the order given. The specific activity (A_o) of the active 28/35 mesh fraction of the feed was measured at the beginning, after the measurement on the 65/100 mesh size fraction of the product, and at the end. The decay-corrected activities at those times were 246.7, 241.0, and 236.9 cpm per gm. The weight (W_o) of the active 28/35 mesh fraction in the feed was 55.0.

[b] Example of calculation for P in the 65/100 mesh product fraction; $A = 29.6$, $W = 45.7$, $A_o = 242.7$, $W_o = 55.0$: $P = \frac{A}{A_o} \text{ x } \frac{W}{W_o} = 0.102 = 10.2$ pct.

A relatively simple procedure was developed by which albite, containing sodium, was activated in the M.I.T. cyclotron. The cyclotron makes high-speed deuterons which impinge on a beryllium target, thereby producing a concentrated neutron flux. The mineral was exposed to this flux for 2 hr. This treatment changed enough of the sodium to sodium 24 (14.8 hr half-life, 1.4 mev β) as to make detection and measurement easy. The nuclear reactions taking place were:

$$_{11}\mathrm{Na}^{23}\ (\mathrm{n}, \gamma)\ _{11}\mathrm{Na}^{24} \text{ (irradiation)}$$

$$_{11}\mathrm{Na}^{24} \xrightarrow{\beta, \gamma, \gamma} {}_{12}\mathrm{Mg}^{24} \text{ (decay)}$$

The detailed technique of the experimentation was as follows: 40 kg of hand-sorted, lump albite were crushed to pass 10 mesh. After careful mixing of the lot, a screen analysis was made. The whole lot of material was fractionated on standard Tyler screens from 14 down to 200 mesh. Samples for experiments were compounded from these fractions in accordance with the screen analysis. When it was desired to make an experiment in which, for example, the 28/35 mesh size fraction was to be studied, the blend of size fractions was made as indicated above, except that the 28/35 mesh size fraction was added only after irradiation in the cyclotron.

The blended charge containing the activated albite was ground for 2 min in a laboratory ball mill with a steel ball charge of controlled size distribution. The ground product was carefully sized on a set of Tyler screens in a Ro-tap. Each size was analyzed for radioactivity by the use of an end-window Geiger-Mueller counter and standard scaling circuit. This analysis was carried out in detail as follows: a 20-g sample was placed in a Petri dish, packed carefully to obtain reproducible geometric distribution with reference to the Geiger-Mueller tube, and the activity was counted for a 2-min period.

Several determinations of the activity of the active size fraction in the feed were made at various times to establish the decay in activity with time. Linear interpolation was used to evaluate the activity that the active size fraction in the feed would have had at any given instant. The ratio of the observed activity in a size fraction of the product to the activity that the active size fraction in the feed would have had at the same time gives the fraction in the product size that came from the irradiated size in the feed.

The general formula for finding the distribution, P, of a specific individual size fraction in the feed

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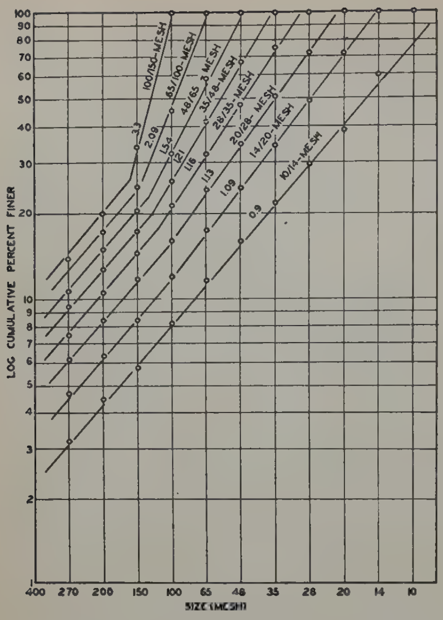

Fig. 1—Progeny of feed size fractions from 10/14 down to 100/150 mesh after 2 min of grinding.

Numbers on curves represent slopes.

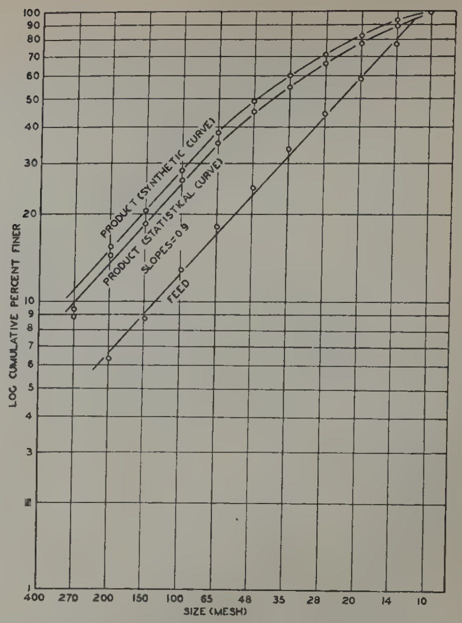

Fig. 3—Size distribution of feed and of product after 2 min of grinding.

into some particular size fraction of the product is:

$$P = \frac{A}{A_o} \times \frac{W}{W_o}$$

In this formula A and W are the specific activity and weight of the size fraction of product on which interest is momentarily focused, and A_o and W_o are the corresponding decay-corrected specific activity and weight for the active size fraction of the feed. As a typical example, reference may be made to Table I, which gives the results for one particular experiment in which the 28/35 mesh fraction was irradiated.

Study of grinding of the same feed, but with attention focused on parent particles of different sizes was made in a similar fashion. The data for all the experiments are presented in Fig. 1. This figure shows the progeny of particles of all sizes from 10 to 14 mesh down to 100 to 150 mesh. The line for the coarsest feed size (10 to 14 mesh) is like the lines obtained when a feed of one size is crushed or ground. But the lines for the finer feed sizes (say 65 to 100 mesh) suggest a behavior not recognized until now. Specifically, it suggests that fracturing one size in a normal environment of all sizes may produce relatively fewer fines than is indicated by a slope of one in Fig. 1. This is what is wanted in practice. If it could be shown that the phenomenon is general and how advantage could better be taken of it in practice, a new impetus would be given to utilitarian comminution studies. The experiments deserve checking before indulging in speculation.

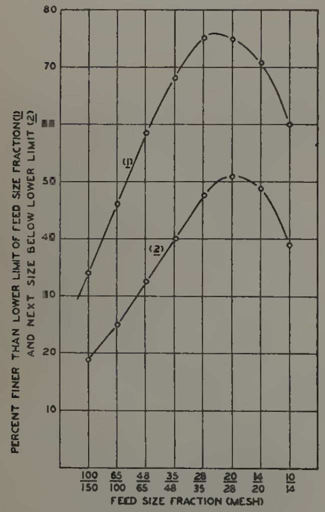

Fig. 2—Extent of size reduction for several feed size fractions after 2 min of grinding.

It is also of interest to observe that the proportion of the initially marked material that is reduced by more than a factor of $\sqrt{2}$ or a factor of 2, or any factor selected, is not constant with size but depends characteristically on it. This observation is brought out by Fig. 2. To what extent this is caused by crushing action and to the harmony between the size distributions of the feed and the grinding device selected are matters which may only be guessed at, at present.

The overall size distributions of the feed and product are found in Fig. 3. The synthetic product curve was obtained by summing up the size distributions of each size fraction after grinding and converting these results to cumulative percent finer. It is to be noted that the synthetic curve follows the actual curve as to shape but is slightly displaced. The displacement was possibly caused by the experimental error introduced by stopping the screen analysis at 400 mesh. In spite of this variation, the agreement between the curves as to slope and shape indicates that the experimental technique is valid.

The method of radioactive marking as a crushing research tool is both simple and effective. It seems that the method can be extended considerably. For example, an unusual constituent may be introduced in radioactive form in mixed feeds. Many of the variables that have been baffling in comminution operation such as comparison between crushers, pulp density, speed of mill, preferential grinding of a given mineral, etc., might also be attacked with this new tool. Finally, the method seems to offer great promise in dealing with problems involving circulating loads where there is really no suitable commonplace technique now available.

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Jaw Crusher Capacities, Blake and Single-Toggle Or Overhead Eccentric Types

by D. H. Gieskieng

THE advent of curved jaw crusher wearing plates made an approach other than segmental layout analysis desirable for prediction of capacities. For some time it had been known that the drawing board capacities of crushers using these plates had to be considerably modified by complicated experience factors to achieve agreement with results. Because these apparent capacities could be readily increased severalfold by minor crushing chamber shape changes, it was necessary that the utmost precaution be taken in predicting capacities of jaw plates modified for nonchoking, special wear characteristics, or any other reason.

To this end the laboratory and field tests outlined by the author in a previous paper[1] were made on Blake-type jaw crushers. The results of these tests were summarized in a simple first degree equation applicable to crushers using either straight or curved jaw plates. This equation first outlines the maximum capacity potential of a given crusher, then reduces this figure in accordance with installation circumstances by means of a realization factor.

It was found subsequently that this equation, with the addition of an eccentric throw factor, is applicable to standard types of single-toggle or overhead eccentric jaw crushers as far as maximum capacity potential is concerned. However, these crushers have realization factor curves somewhat different from those outlined for the Blake type.

While this paper is concerned principally with standard type single-toggle crusher capacities, the evaluation of data obtained with these machines is simplified by comparative reduction to the 10 x 7 in. Blake-type equivalents upon which the summary of the preceding paper was made. Convertibility of data from one type of crusher to the other also tends towards confirmation of both. The agreement of these data is sufficient to be considered complimentary. Consequently the feed factors, f, previously reported for Blake crushers are slightly adjusted to an average with the single-toggle crusher results.

Blake-type equation:

$$C = f\cdot d\cdot w\cdot y\cdot t\cdot n\cdot a\cdot r \qquad [1]$$

Single-toggle type equation:

$$C = f\cdot d\cdot w\cdot y\cdot t\cdot n\cdot a\cdot e\cdot r \qquad [2]$$

where C is the capacity in short tons per hour through the crusher, f is a feed factor, dependent upon the presence of fines in the feed, and the surface character of the jaw plates used.

Values of f:

	Smooth Plates	Corrugated Plates
With normal fines	0.0000414	0.0000319
Fines scalped out	0.0000368	0.0000252
Large pieces only	0.0000312	0.0000215

d is the apparent density of the broken product in pounds per cubic foot. (If the true specific gravity of the feed is known, 40 pct voids may be assumed and d becomes 37.4 times sp gr).

w is the width of crushing chamber in inches.

y is the openside setting of the crusher, in inches. In the case of corrugated jaw plates it is measured from the tip of one corrugation to the bottom of the valley opposite.

t is the length of jaw stroke in inches at the bottom of the crushing chamber. It is the difference between open and close-side settings.

n is rpm, or crushing cycles per minute.

a is the nip-angle factor. It is unity for 26° and 3 pct greater for each less nip-angle degree. A nip-angle of 20° has an a value of 1.18, and an angle of 30° has an a value of 0.88, see Fig. 1.

r is the realization factor. It is unity for perfectly uniform choke feeding and usually less for actual operating conditions according to the method of feeding used and the probabilities of hang-ups involving the size of feed and crusher opening. Approximate values are given by the curves in Fig. 2. These values are further reduced by intermittent feeding.

e is the throw or diameter of gyration of the single-toggle crusher eccentric in inches.

As evident in Fig. 1A, variation of feed size will generally have little effect on nip-angle if both jaw plates have flat areas.

Jaw plates having continuous curvature, as in Fig. 1B will have different nip-angles, depending upon the size of feed. For test work as described in this paper this effect was accounted. For general compilation of capacities for average feeds it is suggested that the nip-angle be taken at the various settings computed, at an arbitrary level, such as is indicated in Fig. 1C.

Data Evaluation

To bring the Blake and single-toggle type crusher capacity test results to common terms for evaluation, all data are converted to terms of 10 x 7 in. Blake-type performance at conditions of 100 lb per cu ft, 10 in. chamber width, 250 rpm, 0.65 in. stroke, 3-in. openside setting, and 18° nip-angle. (The nip-angle of the 10 x 7 in. Blake is 18° at 3-in. setting.) The single-toggle crusher performances are also divided by the eccentric throw to bring this effect to unity.

As outlined,[1] laboratory and field tests made on Blake-type crushers ranging from 10 x 7 in. to 60 x 48 in. were summarized along the foregoing conditions of speed, stroke, etc. This resulted in groups of data which correspond to feeds with fines, feeds

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with fines scalped, and feeds consisting of large pieces only. The results of this Blake-type summary are averaged with the single-toggle type data as converted to Blake-type equivalents, and this average forms the basis for calculation of the revised feed factors, f, i.e., considering corrugated jaw plates and feeds with fines.

$$f = \frac{19.3}{100 \times 10 \times 3.0 \times 0.65 \times 250 \times 1.24} = 0.0000319$$

where the numerator is the 10 x 7 in. Blake-type equivalent tonnage and the denominator corresponds to density, width, setting, stroke, speed, and nip-angle at the conditions which produced this tonnage.

Crushability-Capacity Effect

In the Blake-type crusher tests,[1] no capacity variation was noted for materials of different crushabilities, even though a wide range of materials was tested. These feeds had impact strengths ranging from 2.8 to 31 ft lb per in. of thickness as measured by the Bond method,[2] (potash, coke, soft hematite, limestones, traprock, taconites.)

The single-toggle crusher tests upon which this present paper is based indicate a trend in crushability-capacity effect. From Table I, which gives single-toggle to Blake-type equivalents, it is evident that feed *A*, a relatively soft gravel, resulted in capacities about 10 pct higher than those obtained with considerably tougher feeds *B* and *C*. A few tests not listed were run with very friable dry bituminous coal, which further indicated a crushability-capacity trend for single-toggle type crushers.

The simplicity of the capacity equations is maintained without loss of practical accuracy by averaging the single-toggle crusher results obtained with tough feeds (*B*, *C*, and *D*), and average feeds (*A*). It is evident that very little error is introduced for most feeds by doing this (5 pct or less). If softer than average feeds are contemplated for single-toggle crushers, up to about 10 pct additional capacity might be expected.

It is believed that the single-toggle crusher crushability-capacity trend is largely caused by the eccentric action which results in a rubbing motion between the jaw plates (attrition). With tough feeds this effect is apparently negligible as far as capacity is concerned. Feed factors computed from the traprock and taconite tests came within about 3 pct of feed factors previously reported for general feeds with Blake-type crushers.

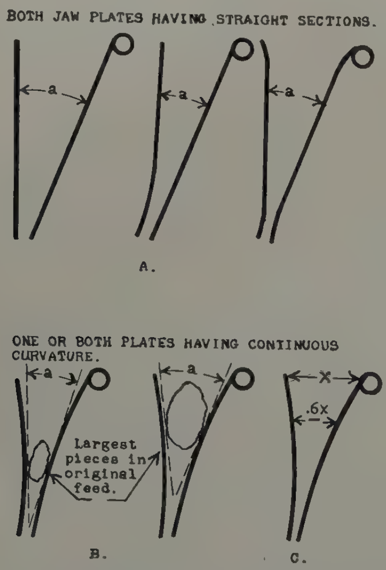
BOTH JAW PLATES HAVING STRAIGHT SECTIONS.
a
a
a
A.
ONE OR BOTH PLATES HAVING CONTINUOUS CURVATURE.
a
a
x
.6x
Largest pieces in original feed.
B.
C.

Fig. 1—Location of nip-angle measurement in Blake-type or single-toggle type jaw crushers.

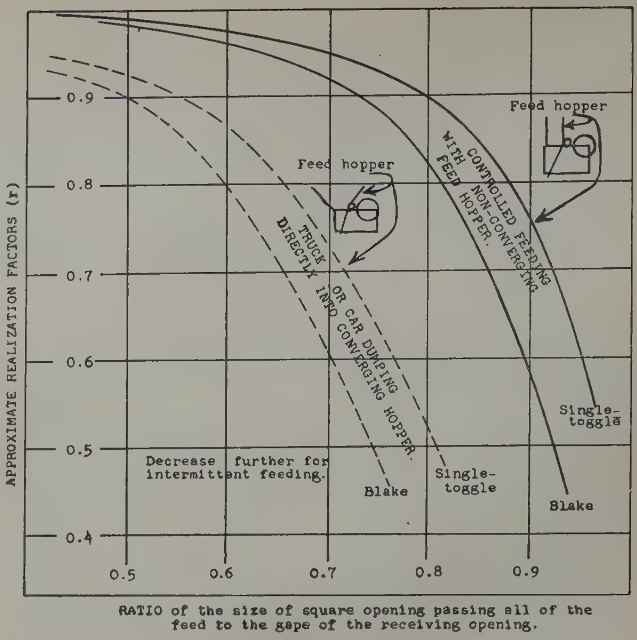

Fig. 2—Realization factors (r).

Greater differences in crushability-capacity effect than those just discussed for single-toggle type crushers have been reported by investigators working with small Dodge-type crushers. However, these crushers have rubbing motion between the jaws at the discharge, Fig. 3, and in addition have very little jaw stroke at the discharge. The crushing done by attrition between the jaws thereby assumes increasing importance with more friable feeds, as there is longer retention time in the crushing chamber caused by the small stroke and resulting discharge capacity deficiency. Comparing various data reported for Dodge-type crushers, it was found that the so-called crushability-capacity effect was very much greater with a jaw stroke of 0.04 in. than it was with 0.20 in. A further contributing effect to this tendency is believed to be illustrated in Fig. 4 of the preceding paper[1] which indicates that discharge capacity falls off more than proportionately for jaw strokes less than about ¼ in.

Reduction of data from one type or size of jaw crusher into the equivalent performance of another crusher has been accomplished by ratios of all of the various equation factors involved. The individual

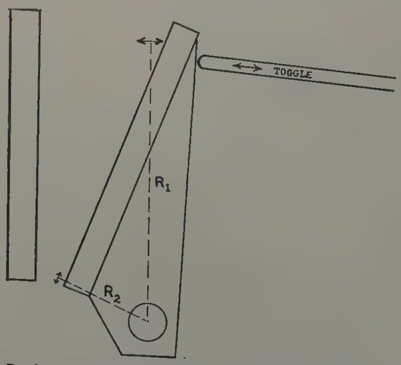

Fig. 3—Dodge-type jaw crusher showing relative swing jaw motion at feed and discharge ends.

Table I. Single-toggle Data Conversion to Blake-type Comparison Equivalents

Single-Toggle Crusher Size and Eccentric Throw, In.	Open and Closed Settings	Nip-angle	10x7 in. Blake-type comparison equivalents. 100 lb, 250 rpm, .65 in. stroke, 3 in. openside setting, 18° nip-angle, 10 in. width	
CORRUGATED JAW PLATES				
Feeds with Fines				
4¾ x 3⅛ ½ ecc.	0.932/0.616	16.7°	19.8 A	18.8 B (2)[a]
	0.563/0.250	19.2°	21.0 A (2)	19.5 B (2)
6⅜ x 5⅝ ¾ ecc.	1.385/0.817	19.4°	21.2 A	20.4 B
	1.037/0.470	20.9°	21.9 A	18.5 B
			20.1 Avg 18.5 Prev. Blake	
			19.3 Avg Feed factor 0.0000319	
Feeds with Fines Scalped				
4¾ x 3⅛ ½ ecc.	0.938/0.622	16.6°	16.6 A	16.3 B
	0.563/0.250	19.2°	14.6 A (2)	13.9 B (2)
6⅜ x 5⅝ ¾ ecc.	1.250/0.676	20.0°	16.1 A	13.2 B
	1.000/0.435	20.9°	16.7 A	14.1 B
	0.846/0.278	21.7°	13.6 A	13.1 B (2)
36 x 25 1¼ ecc.	3.91/3.04	26.0°		15.4 C
			15.0 Avg 15.4 Prev. Blake	
			15.2 Avg Feed factor 0.0000252	
Feeds Consisting of Large Pieces Only				
6⅜ x 3⅛ ¾ ecc.	1.715/1.147	17.9°	13.9 A (3)	13.0 B (2)
			13.5 Avg 12.5 Prev. Blake	
			13.0 Avg Feed factor 0.0000215	
SMOOTH JAW PLATES				
Feeds with Fines (—2.25 In. Slot Size)				
24 x 10 ¾ ecc.	1.64/1.10	7.5°	24.4 D	
	1.19/0.65	12.0°	24.0 D	
	0.79/0.25	12.5°	22.8 D	
			23.7 Avg 1.2[b]	
			24.9 Avg 25.0 Prev. Blake Feed factor 0.0000414	
Feeds with Fines Scalped (—3 In. Slot Size)				
24 x 10 ¾ ecc.	1.64/1.10	11.0°	21.9 D	
	1.19/0.65	12.5°	21.8 D	
	0.79/0.25	14.5°	20.3 D	
	Density	**Impact Strength**	21.3 Avg	
Feed A gravel	105 lb per cu ft	15.0	1.1[b]	
Feed B traprock	105 lb per cu ft	27.0	22.4 Avg	
Feed C taconite	130 lb per cu ft	20.3	22.0 Prev. Blake	
Feed D traprock	107 lb per cu ft	23.7	22.2 Avg Feed factor 0.0000368	

[a] (2), (3) indicates average of two or three tests.

[b] 5 pct added to compensate for average feeds not tested.

analysis of the extent of most of these factors is outlined in the preceding paper.[1]

Direction of Flywheel Rotation

The top of the flywheels of almost all single-toggle crushers rotate towards the crushing chamber, and this rotation is considered to be normal. Capacity eqs 2 and 3 and subsequent discussion are based upon this rotation unless otherwise noted.

Tests made with reverse rotation on standard and inverted toggle single-toggle crushers indicate about 20 to 30 pct less capacity for average crushing conditions and nip-angles of about 20°. With small ratios of reduction or larger nip-angles the capacities obtained with reverse rotation approach those obtained with normal rotation.

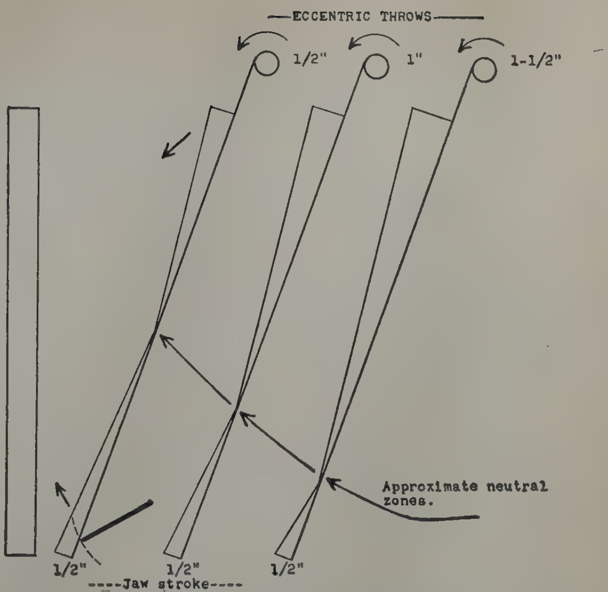

Fig. 4—Force feeding and uplifting action in single-toggle crusher.

A single-toggle crusher with a 26° nip-angle had 6 pct more capacity with normal rotation than with reverse when crushing normal feeds. However, in the particular pit where this machine was located, a large portion of stream-worn feed was present of a size corresponding to the neutral zone, Fig. 4. These boulders handicapped the crusher to such an extent that reversing the rotation and thereby dislocating the neutral zone improved the overall operation. This is believed to be an exception.

Toggle Arrangements

Standard: Almost all single-toggle crushers built today are arranged with the toggle slanting downward to the swing jaw, see Fig. 5a. This arrangement is considered standard and lends itself to strong construction as the toggle is in compression and the pressure reaction is in line with the crushing chamber.

In analyzing the jaw motion of these crushers, it may be seen that during about 50 pct of the time the opening and closing motion at the top and bottom of the crushing chamber is opposite. This results in a variable intermediate level having little crushing action, which may explain to some extent the applicability of the capacity equation to either the standard type of single-toggle or Blake-type crusher. This characteristic has some design advantage for choke feeding as the flow of material to the lower portion of the crushing chamber is limited, which tends to reduce packing tendencies in this critical region.

Inverted Toggle: Single-toggle crushers having an opposite toggle action to that of the so-called standard type are for convenience referred to as Inverted toggle crushers, Fig. 5b. This type is uncommon and the few known in the field are old.

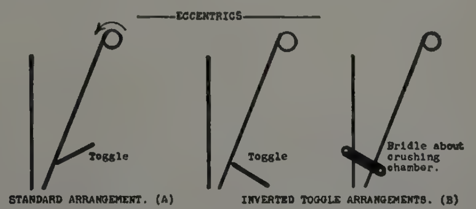
ECCENTRICS
Toggle
Toggle
Bridle about crushing chamber.
STANDARD ARRANGEMENT. (A)
INVERTED TOGGLE ARRANGEMENTS. (B)

Fig. 5—Standard and inverted toggle arrangements in single-toggle crushers.

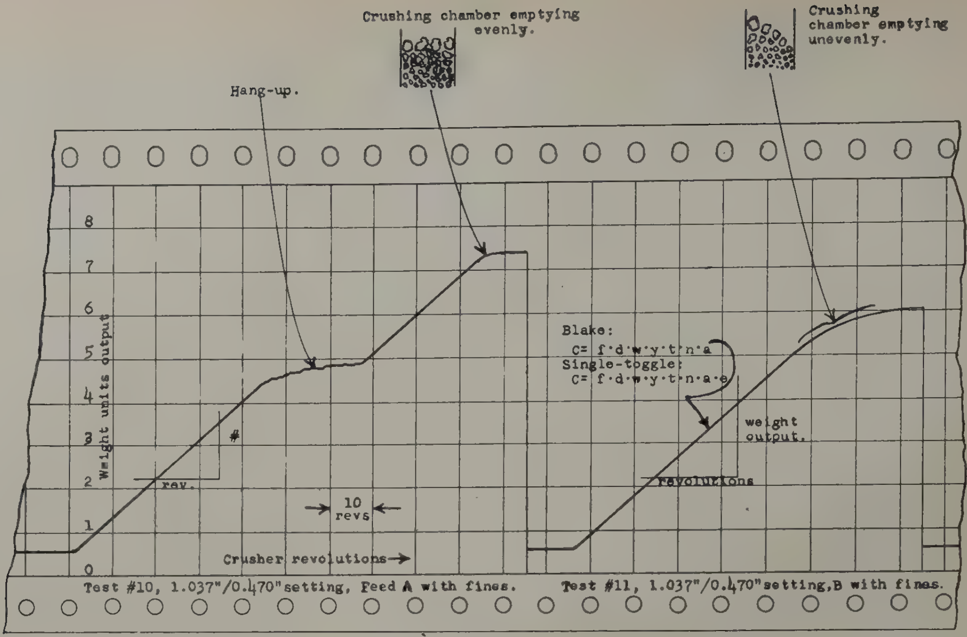

Fig. 6 — Typical crushing test records made by synchronized recording device. Power input measured separately.

An inverted arrangement may consist of either a conventional toggle in compression arranged to slope upwards to the swing jaw or of a bridle consisting of hinged tension rods on both sides of the crushing chamber. The construction of either inverted toggle arrangement is not inherently as strong as the standard type. Also, the feed to the lower chamber tends to be excessive which necessitates relatively small strokes to avoid compaction of the feed and high crusher stresses.

An inverted toggle crusher was tested to round out the single-toggle investigation. The capacity characteristics of this crusher were found to be somewhat different than the standard type of single-toggle crusher. A preliminary capacity equation based upon these tests is as follows:

$$C = f \cdot d \cdot w \cdot y \cdot (t_1 + t_2)\ n \cdot a \cdot r \qquad [3]$$

where t_1 is the stroke at the discharge, and t_2 is the stroke at the top of the crushing chamber.

Eccentricity

Single-toggle crushers with normal flywheel rotation have a crushing action caused by the eccentric motion which is commonly termed forced feeding. With standard single-toggle crushers this crowding action extends only to the previously mentioned variable intermediate level having little crushing motion. Below this level an opposite or uplifting action takes place.

The observation that a force feeding action is capacity conducive, and that an uplifting action is not, is confirmed by the results obtained in the reverse rotation experiments. It may be seen that balancing of the effects of these two actions, as illustrated in Fig. 4, is largely accomplished by the presence of the factors e and t in eq 2; as the value of e increases the proportion of uplifting action decreases.

Since standard type single-toggle crushers having eccentric throws of ½ in., ¾ in., and 1¼ in. were tested, and the results successfully converted by ratios of these eccentricities, this eccentricity factor is apparently linear for all practical purposes.

In converting standard type single-toggle data to Blake-type equivalents, the former is divided by the eccentric throw to reduce the effect described above to unity.

Realization Factor

By means of apparatus which continuously recorded the crusher tests, it was possible to eliminate the vagaries of feeding conditions as illustrated in Fig. 6, and thereby reduce the first analysis to terms of characteristic maximum capacities at given conditions of setting, stroke, etc. Various field data were then compared to the resulting equations to determine approximately what percentages of the potential capacities were obtainable in practice with various feeding methods. The effect of various feeding methods is included as a realization factor, r.

Summary

This paper is concerned with the capacity characteristics of jaw crushers, and no attempt is made to discuss the other characteristics, such as power consumption or screen analysis of product. It is assumed that in applying the equation good installation practice is followed to the extent of scalping the feed when small settings are used and employing jaw plates having a proper nonchoking curvature if feeds having unusual packing tendencies are encountered.

Acknowledgment

The writer expresses appreciation for the fine cooperation of his associates in the company and those in the field, who from the first appreciated the possibilities of the investigation and gave every assistance towards its completion. Particular acknowledgment is given Bruce Irwin, formerly of the Allis-Chalmers Basic Industries Laboratory, who supervised the tests on the 24x10 in. single-toggle crusher.

The opinions expressed on the controversial subject of crushing are those of the writer and do not necessarily coincide in all particulars with those of the Allis-Chalmers staff.

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Use of Isopachous and Related Maps in the Florida Phosphate District

by Thomas E. Wayland

AN isopachous map is one on which lines connect points of equal thickness of a given unit. This type of map is used by the Florida Phosphate Project of the U. S. Geological Survey to represent the economic phosphate deposits known as matrix and the waste material, or overburden, that overlies the matrix.

The top of the bed on which the phosphate was deposited is known as the *basement* and a subsurface contour map of this old buried erosion surface is known as a basement map. Recent experiments have been made in preparing maps that show tonnages and grades of the phosphate content of the matrix.

Few of the operating companies in the Florida phosphate district have applied isopachous (Greek *isos,* equal and *pachys,* thick) to mapping. The writer believes there is a need for the techniques discussed herein and that they can be applied to mapping other geologically similar areas in either economic or scientific investigations.

The land-pebble phosphate district of Florida occupies a compact area in the west-central part of the state. It includes mainly the following land survey divisions: Ts. 27 S. through 32 S. and Rs. 20 E. through 26 E. The town of Mulberry, Fla., is in the approximate center of the district.

The strata of the area, which is part of the Gulf Coastal Plain, occur in thin formations with broad outcrop belts, and low dips. The topography is subdued and gently rolling with three marine terraces, which are found at 30, 100, and 150 ft above sea level,[1] accounting for most of the relief. Occasional small sinkhole lakes are present, most of them above the 150-ft shoreline.

The phosphate deposits occur in unconsolidated sediments such as clays, sands, and sandy clays. They are overlain by a heterogeneous assemblage of sands, clays, muck, and iron-cemented sand, easily penetrated, in most cases, by a hand auger or drill. Limestone, locally called bedrock, or a calcareous bedclay, thought to be a residue of the limestone, directly underlies the phosphate deposits.

General Requirements

Most companies and independent prospectors operating in the district have furnished prospecting data to the U. S. Geological Survey. The information is recorded on either field logs or prospecting maps and includes the following information for each hole drilled: location of the hole, thickness of the overburden, thickness of the matrix, phosphate content in long tons per acre, grade of the phosphate content expressed as the percentage of bone phosphate of lime (P_2O_5 x 2.18) or BPL, and the percentages of iron-aluminum oxides and insolubles. The phosphate is classified according to size as either pebble or flotation material. The milling processes of the companies vary, and the size classification is necessarily different in many cases. However, pebble may be considered as larger than 14 mesh and flotation material as smaller than 14, but larger than 150 mesh. Some prospecting data include the exact depth at which bedrock or bedclay was reached, and these figures greatly increase the reliability of the data both for isopachous mapping and for mapping the basement.

A drilling density of four holes per 40 acres of land furnishes a minimum amount of data for isopachous and related mapping. From the minimum of four, densities up to 32 holes per 40 acres are used. The various drilling densities may influence the choice of the proper scale.

Selection of the proper scale is dependent upon the known drilling densities, the subsurface variations to be shown, the extent of the area to be mapped, and the detail desired in the completed map. Scales of 1:24,000, 1:4800, and 1:2400 are used in isopachous and related mapping by the Florida Phosphate Project.

The 1:24,000 scale is used most effectively with drilling densities not exceeding eight holes per 40 acres. The subsurface variations should be relatively low and uniform, permitting the use of smaller intervals without undue crowding of the lines. Comparatively large areas can be mapped on this scale, but minute detail is necessarily sacrificed, because the information is drawn from a maximum drilling density of only eight holes per 40 acres.

Isopachous and related maps of the 1:4800 scale are made of areas on which the drilling information covers from 4 to 16 holes per 40 acres. Moderate subsurface variations with relatively sharp gradations can be shown accurately. The area represented by the maps is reduced considerably in favor of detail.

The 1:2400 scale is most frequently used by the Florida Phosphate Project. It lends itself particularly well to isopachous and related mapping, being easily adapted to the multiform drilling data available. Maps of this scale are prepared with information ranging from 4 to 32 holes per 40 acres; however, use of the minimum drilling density on the

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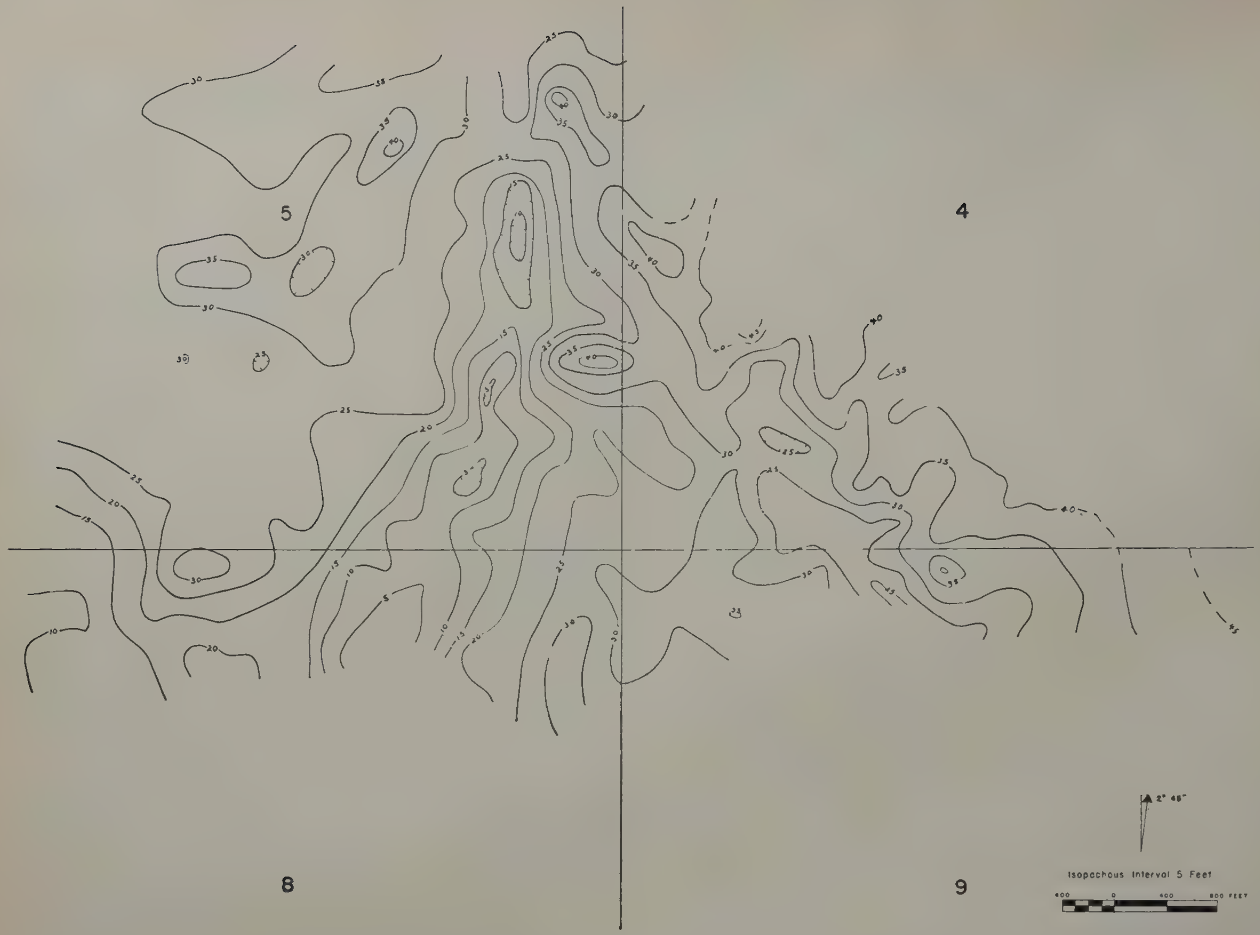

Fig. 1—Overburden isopachous map, Old Colony Mine, Polk County, Fla.

Sections 4, 5, 8, 9, T. 32 S., R. 26 E., American Cyanamid Co. The term *overburden* includes all the material overlying the phosphate deposits as well as the upper, uneconomic part of the deposit. Data from company prospecting, 1935.[2]

1:2400 scale is restricted to areas that bound or enclose adjacent areas having drilling densities of 8, 16, or more holes per 40 acres. Maps of this scale that are prepared from a minimum amount of information should be regarded as tentative, pending the availability of more complete data; on the other hand, these maps are often necessary for completeness and may show at least an outline of a pattern or trend of the strata or deposit if such should be present. Sharp gradations in the character of the subsurface are mapped conveniently on this relatively large scale. The area represented by the map is sacrificed further to show detail.

Equal thicknesses, equal altitudes, and equal tonnages are represented by lines drawn on the isopachous, basement, and tonnage and grade maps, respectively. The relatively large scales, usually uniform thicknesses of strata, and sufficient drilling densities favor a 5-ft interval in most isopachous and basement maps; however, if more detail is desired, a 1-ft interval may be used, depending upon the subsurface variations, drilling densities, and scale.

The subdivision of land in the land-pebble phosphate district of Florida is based on the Congressional Land Survey System of townships, ranges, and sections. The isopachous and related maps of the district have been drawn on plats and base maps with the section as the primary division.

A grid system of east-west, north-south coordinates, used locally by the phosphate industry, has been adopted for use in isopachous and related mapping of the Florida Phosphate Project. Sections are divided into 2½-acre squares by the coordinates, using the numbers 1 through 16 from south to north and the letters A through P from west to east. Drill holes are located usually in the center of the 2½-acre squares, although there are occasional half-line holes, or holes that fall on the lines of the grid pattern. The half-line holes are used mainly for drilling an excess of 16 holes per 40 acres.

Isopachous Maps

Maps representing equal thicknesses of both the overburden and matrix have identical modes of construction. A separate map is prepared for each area on which prospecting data are available.

Overburden and Matrix: Thickness figures are plotted in their respective positions on the base maps or plats. Lines are drawn to connect the points of equal thickness, the spaces between the lines being determined by interpolation, see Figs. 1 and 2. The isopachous maps of the overburden and matrix[2] show at a glance both the depths to which the dragline operator should remove the material and the approximate areas in which he should operate. Consistent with the nature of all maps, these may not be accurate in minute detail, but they can be of aid in determining the approximate points at which changes can be expected. Superimposition of the overburden map on the map of the matrix may reveal a correspondence of either relatively thick or

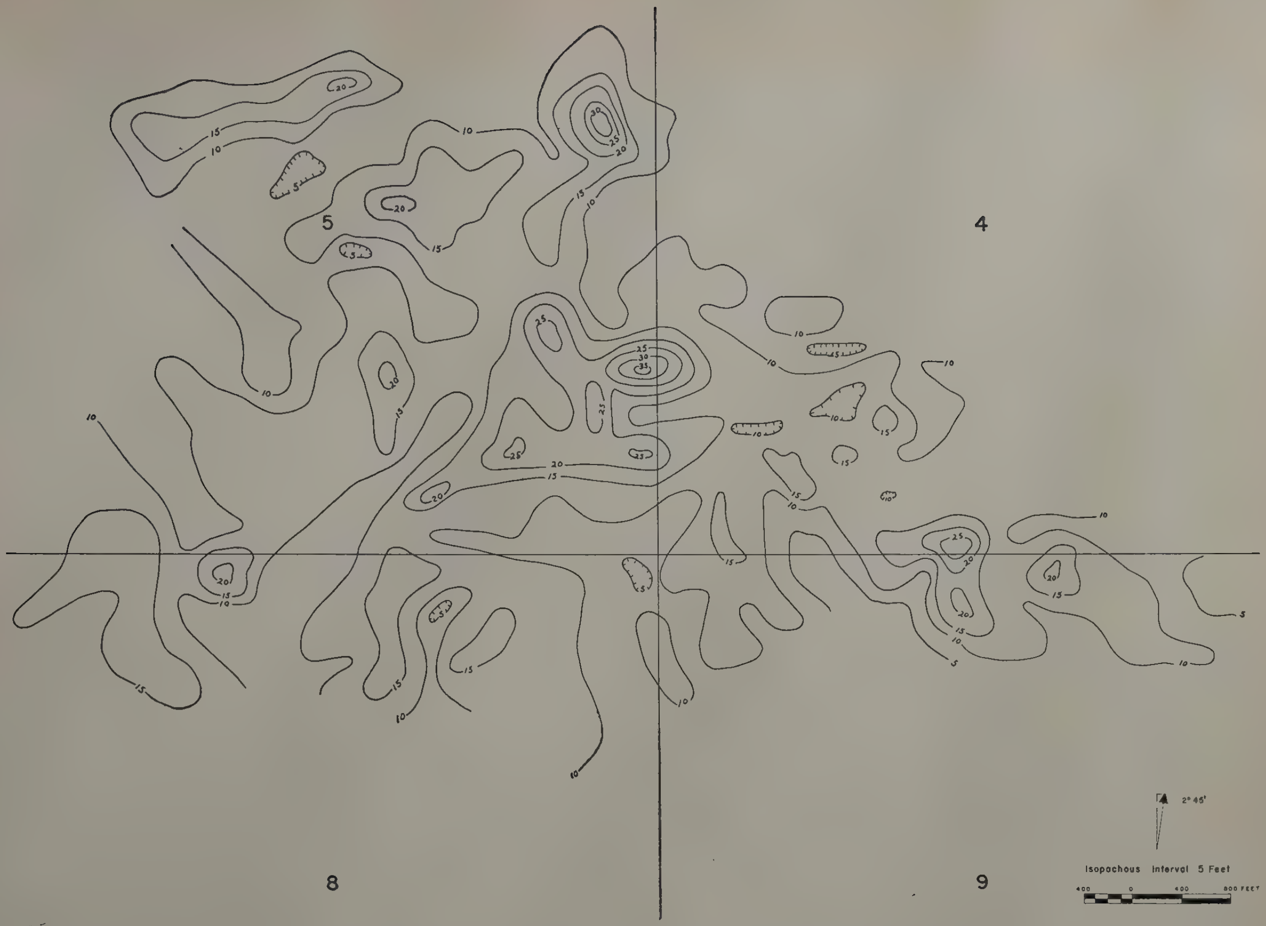

Fig. 2—Matrix isopachous map, Old Colony Mine, Polk County, Fla.

Sections 4, 5, 8, 9, T. 32 S., R. 26 E., American Cyanamid Co. ***Matrix*** **is the term used locally to designate the economic part of the deposit. Data from company prospecting, 1935.**[2]

thin areas of overburden and matrix, thereby aiding the mining engineer in the location of minable areas and possibly indicating trends of the deposit applicable to future prospecting.

Some trends of scientific interest have been noted by the members of the Florida Phosphate Project engaged in isopachous mapping. These trends are suggestive of old buried stream channels and sinks. Similar maps prepared from drilling information predicated on geologic rather than economic requirements would aid in determining the approximate thickness of subsurface formations and in plotting their lateral extent.

Basement Maps

Known points of altitude on the surface immediately over the area to be mapped are required for construction of the basement[1, 2] map. Prospecting data that include the exact depth at which bedrock or bedclay was reached are the more desirable. If these figures are not given, it must be assumed that the prospector did not include bedclay thicknesses with those of the matrix.

The total depths of the drill holes (sum of the overburden and matrix thicknesses) are subtracted from the surface elevations. The resultant figures are the altitude of points at the top of the bedrock or bottom of the matrix. Subsequent plotting of these figures on a base map or plat, and the connection of points of equal altitude by contour lines, complete the basement map, see Fig. 3.

The mining engineer engaged in strip mining may want to know the altitude of the points at the top of the bedrock. A basement map can aid him materially in foreseeing some of the difficulties of drainage during strip mining and in planning his mining operations accordingly.

Tonnage and Grade Maps

The maps of tonnage and grade are prepared from the analytical results included in the prospecting data. One map represents both tonnage and grade.

Tonnage figures are plotted on a base map; lines are drawn to connect the points of equal tonnage, the spaces between the lines being determined by interpolation. An interval is chosen that represents most accurately the tonnages of the particular area being mapped.

Grade figures are now placed in their respective positions along with the tonnage figures. Again, the interval is chosen to give the most accurate representation of the grades, and lines are drawn to connect the points of equal grade. A linear pattern is used to show the areas that fall within the various grade intervals on the map illustrated in this paper, see Fig. 4. The linear pattern was chosen for publication purposes; however, a system of color shading is recommended for practical usage.

The expected recovery of phosphate by tons per acre and by grade can be estimated from a tonnage and grade map. A polar planimeter is used to compute the tonnage and grade of the areas outlined on the map by tonnage lines and grade patterns.

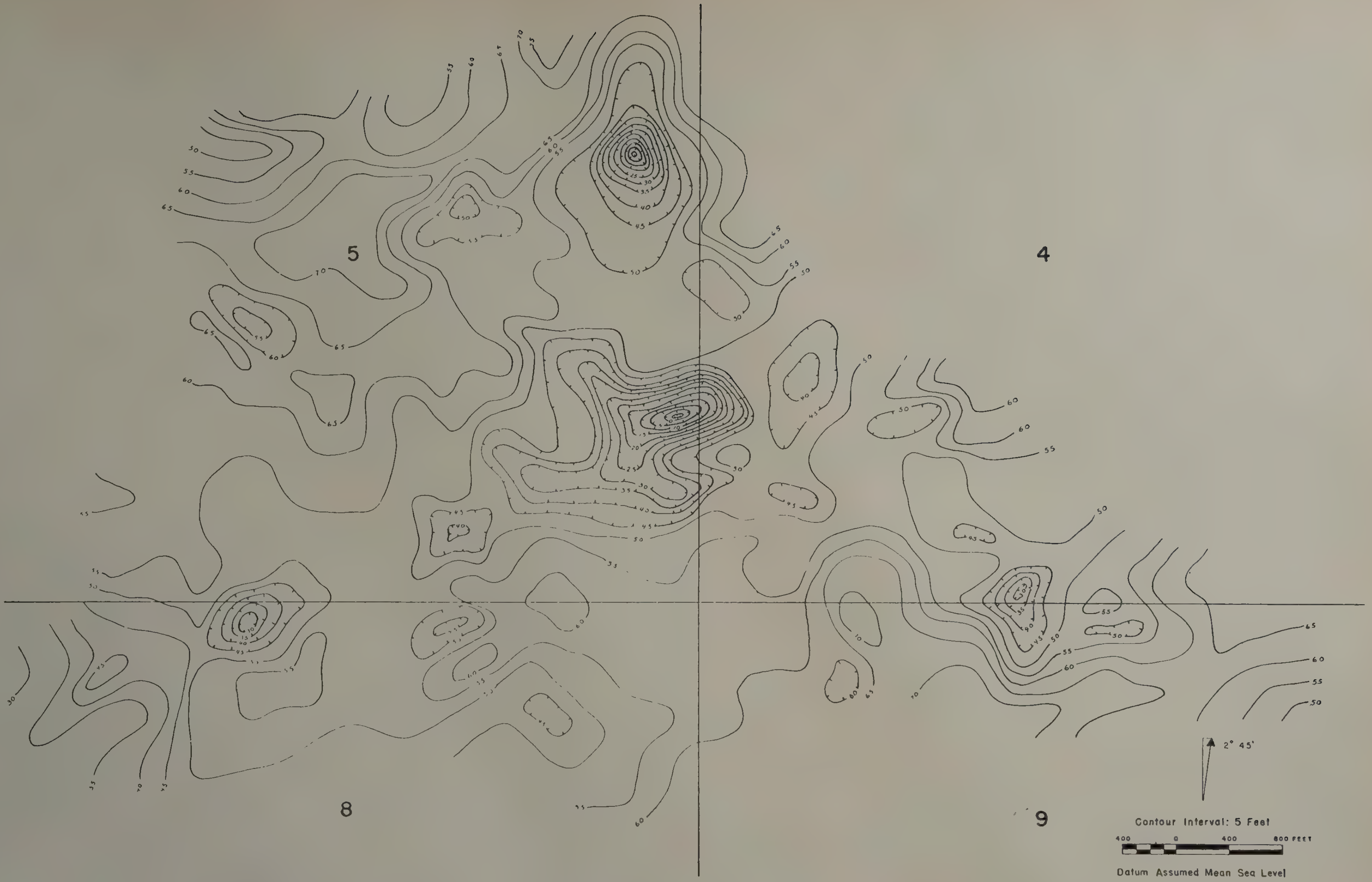

Fig. 3—Basement map, Old Colony Mine, Polk County, Fla.

Sections 4, 5, 8, 9, T. 32 S., R. 26 E., American Cyanamid Co. Contours are on the contact between the matrix and the underlying formation or bed rock. Data from company prospecting, 1935.[2]

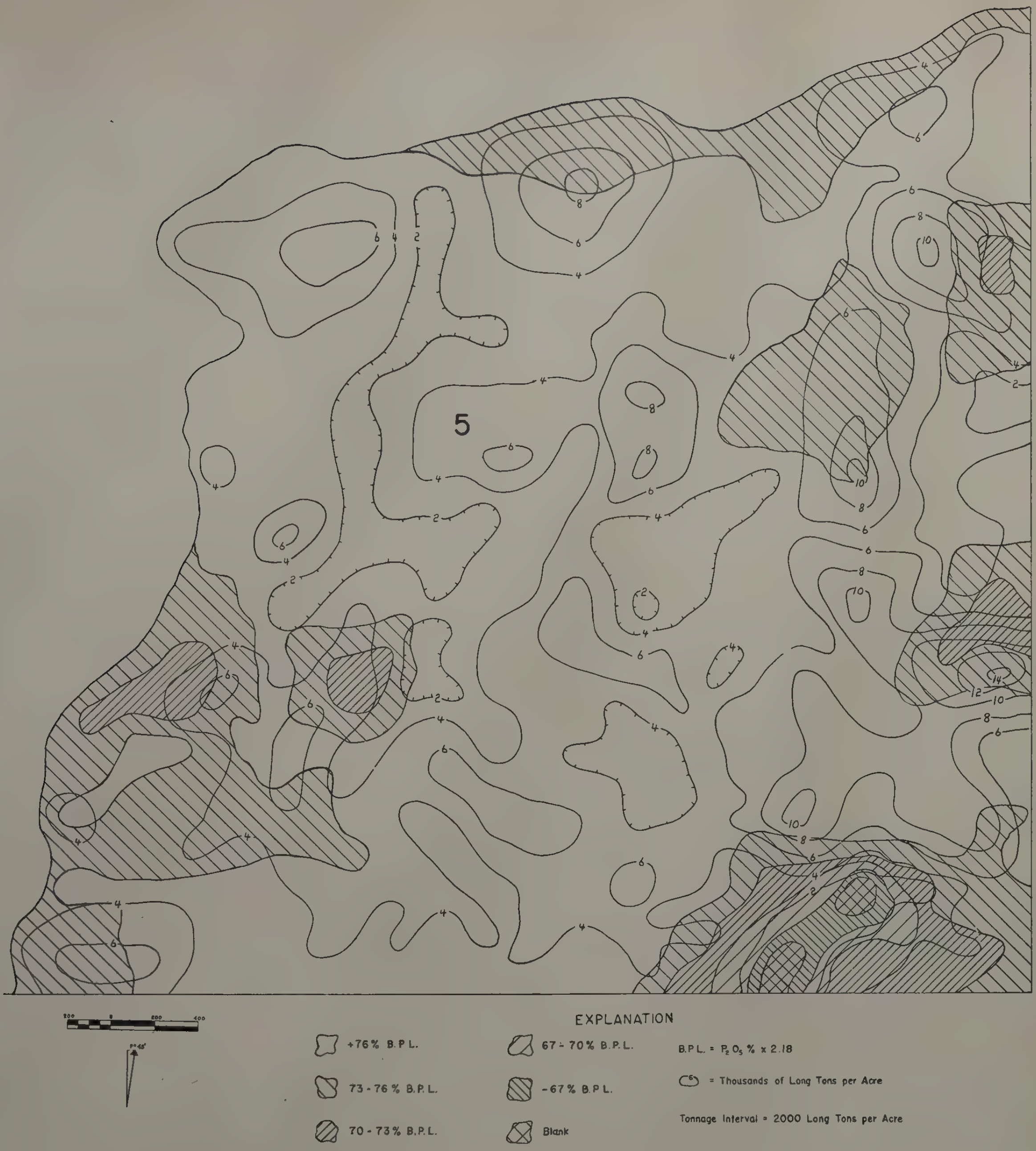

Fig. 4—Tonnage and grade map, Old Colony Mine, Polk County, Fla. Section 5, T. 32 S., R. 26 E., American Cyanamid Co.

Data from analytical results of company prospecting, 1935. Grade interval: grades are divided into 3 pct intervals from less than 67 pct to more than 76 pct BPL

The tonnage and grade map shows clearly the concentration of tonnage and grade for a given area. A tonnage and grade map used in conjunction with the overburden, matrix, and basement maps of the same area may enable the mining engineer to select the best mining location.

Acknowledgments

Owing to the serious illness of Mrs. Maher, Mr. Stuart W. Maher, a coworker of the writer, was unable to participate in writing this paper as was planned originally. The writer wishes to express his regrets of this misfortune and his appreciation of the counsel given by Mr. Maher when the paper was in an early stage of preparation.

J. B. Cathcart, Project Chief of the Florida Phosphate Project, U. S. Geological Survey, supplied the overburden, matrix, and basement maps illustrating this paper and gave valuable criticisms of its content. Arthur Crago and J. L. Weaver of the American Cyanamid Co. made the data on the Old Colony Mine available for publication. The writer would like to acknowledge also the assistance of F. N. Houser, S. L. Houser, H. L. Jicha, Louis Pavlides, R. G. Petersen, and R. H. Stewart, members of the Florida Phosphate Project.

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Discussion[*]

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[*] TP 3193

A — Metal Mining

Rock Hardness as a Factor in Drilling Problems

by W. B. Mather

DISCUSSION

R. G. Wuerker *(University of Illinois, Urbana)*—Mr. Mather is to be congratulated for stressing the most urgent need for a program of testing the physical properties of rocks, as they are encountered by the petroleum engineers on their drilling jobs and by the miners on their excavating, roof control and mineral preparation tasks.

Another valuable source of information is the *Handbook of Physical Constants.*[22] It stresses the physicist's approach rather than the engineer's procedure.

The study of hardness, and rock hardness in particular, is beset with many difficulties. There is no satisfactory definition of hardness yet, and the term hardness is used to describe a number of quite different properties. Hardness tests are grouped into static and dynamic determinations. They should be expressed with reference to the mechanical action of the applied force, like indentation hardness, scratch hardness, abrasion hardness, rebound hardness. A discussion of all these various hardness tests and of the special difficulties encountered in hardness testing of such heterogeneous material as rocks is given in the U. S. Bureau of Mines' standard proposal. The more critical student may be referred to the book by D. Landau: *Hardness,* which is in my opinion the best existing treatise on this subject.

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B — Minerals Beneficiation

A New Surface Measurement Tool for Mineral Engineers

by F. W. Bloecher, Jr.

DISCUSSION

S. Mortsell and **J. Svensson** *(Royal Institute of Technology, Stockholm, Sweden)*—Bloecher states that the apparatus described by him should be a suitable instrument in the mineral dressing laboratories for determining the specific surface of such granular products that are generally investigated.

We want to express a somewhat different opinion, as we doubt that the gas adsorption method of surface measurement really is the most suitable method in mineral dressing laboratories. It is well known that the surface area, as measured by this method, does not coincide with the surface area calculated from the size, shape, and number of particles, i.e. the superficial surface of the particles. It can be seen easily from Mr. Bloecher's own determination on ilmenite-leucoxene concentrate that the surface area measured according to this method may be several hundred times larger than the superficial surface. It is supposed that the surface of cracks, fissures, and pores within the particles also is included in the measured surface. However, this is an assumption which scarcely can be proved and certainly has not been proved.

We believe that the mineral engineer could better use a method giving the superficial surface of the particles, as in most cases this surface only is of importance to him. Such is certainly the case when calculating the resistance of the fluid flow through granular materials and the capillary forces of such materials, problems which are of great importance in connection with dewatering, filtration, and pelletizing[19] etc. The same may also be the case when studying flotation problems, partly since the quantities of the introduced reagents usually are so small that they can only cover the most accessible surface of the particles, and partly since the molecules of at least the collecting reagents are so large that they cannot enter the smallest cracks and pores. As to the surface, which is of consequence in the application of Rittinger's law of crushing, one is inclined to believe, from theoretical reasons and from the work by Gross and Zimmerley, that the total surface here is the most important one. However, it seems as if in that case the superficial surface was as important as the total surface.

Other reasons making the gas adsorption method less suitable for use in the mineral dressing laboratories are that the granular materials there very often are too coarse to be measured by this method, that the experimental technique is rather difficult, and that it takes a long time to measure the specific surface of a sample. Besides, the validity of the method must still be considered questionable. It is thus well known that with different gases one usually gets different values of the specific surface of the same sample, which values, even with such an ideal material as metal foils, can differ as much as about 50 pct.[20] Still more serious

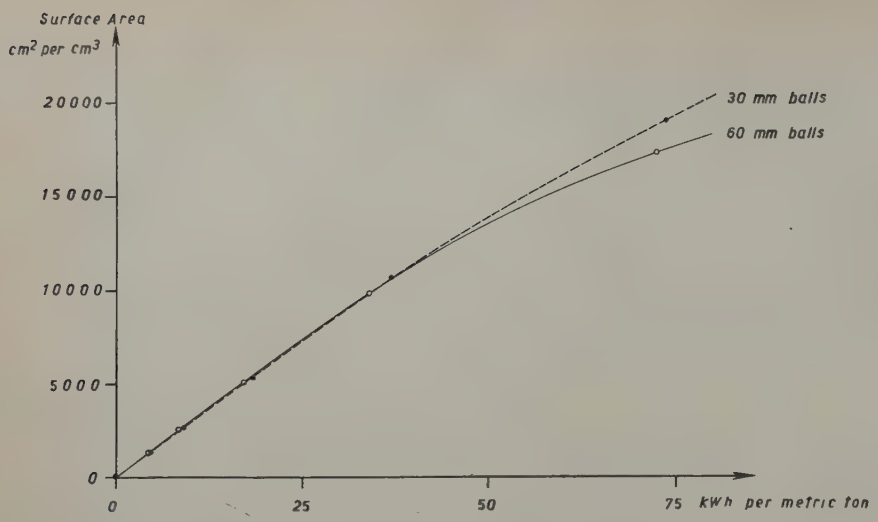

Fig. 1—Specific surface of quartz ground in ball mill as function of the net energy input to the mill.

is perhaps the fact noted by Johnson, Axelson, and Piret that the change of the temperature, from 200 to 550°C, at which their quartz samples were degassed, caused a variation of as much as 50 pct of the values of the measured specific surface.[21]

Instead of the gas adsorption method, we consider the gas permeability method better suited for use in the mineral dressing laboratory, as with that method the superficial surface can be determined. Consequently, it is possible to control and complete the surface determinations made by that method through calculations in those cases where it is possible to determine the particle size distribution of the material and the shape of the particles. For several years we have used this method in the mineral dressing laboratory of the Royal Institute of Technology in Stockholm. Before adopting the permeability method for general use, the validity of the method was thoroughly tested. During the testing it was found necessary to slightly modify the method. The modification mainly consisted in determining the permeability of the bed at several different pressures, so that both the laminar and the molecular components of the gas flow through the bed could be calculated, and in calculating the specific surface from the quotient between these two components.[22] Due to the modification, the determinations take a somewhat longer time (four determinations being made in one day by one man), but instead the specific surface measured in that way agrees closely with the superficial surface of the particles independent of the shape of these or of the porosity in the bed—at least as far as we have been able to control.

Hitherto the method has been used chiefly in connection with work on crushing and grinding conducted within the scope of Jernkontoret's (Swedish Ironmasters Association) research. We hope to publish the complete results of this work later on. However, in this connection it is interesting to note that the results indicate that Rittinger's law, although not generally valid, seems to hold true in the grinding of some homogeneous material as long as the product is not too fine. The results given in Table III from ten dry batch grinding tests will serve as a proof of this. The tests were made with balls of two different sizes on crushed quartz of the size fraction −3.7 +2.3 mm. The results are also plotted in Fig. 1. From the graph it is evident that the specific surface of the grinding product increases in proportion to the calculated energy input up to a specific surface of 5000-10000 cm^2/cm^3, but that the proportionality breaks down at higher values. It might be pointed out that Gross and Zimmerley in their work to verify the validity of Rittinger's law did not go to a higher specific surface than about 2500 cm^2/cm^3.

We think, however, that comparisons between surface areas measured by both the gas adsorption method and the gas permeability method would be very interesting from a theoretical point of view, especially in connection with work aiming at testing the validity of Rittinger's law. For our part, we are willing to cooperate in such a comparison by making determinations of the specific surface area with the permeability method on samples sent to us, the specific surface of which has been determined by the gas adsorption method. We are also willing to place at the author's disposal about a hundred samples, the specific surface of which we have determined by the permeability method, including samples of all the grinding products referred to in Table III.

F. W. Bloecher, Jr. (author's reply)—There are undoubtedly certain problems in which surface area measurements are most conveniently and easily made by the gas permeability method. The krypton apparatus served as a tool in dealing with problems concerning flotation fundamentals. It seems quite logical that any fissure, crack, or pore surface that is measurable by multilayer gas adsorption will also be accessible to any ions or molecules existing in water. Remember that krypton adsorption measurements described take

Table III. Results from Dry Batch Grinding Tests on Quartz with Two Different Sizes of Balls[a]

Size of balls, mm	60					30				
Number of balls	50					436				
Grinding time, min.	12	24	48	96	192	12	24	48	96	192
Number of revolutions	687	1375	2746	5486	10985	687	1372	2749	5480	10968
Energy input to mill motor, w-hr	49	94	191	382	793	50	101	201	404	810
Efficiency calculated	0.50	0.49	0.49	0.49	0.50	0.50	0.50	0.50	0.50	0.50
Net energy, w-hr	24	46	94	187	397	25	50	101	203	405
Net energy, kw-hr per ton	4.36	8.36	17.1	34.0	72.2	4.55	909	18.4	36.9	73.6
Percentage through 200 mesh (74.3μ)	17.4	30.3	51.5	74.0	92.4	18.5	33.5	59.8	86.7	98.8
Percentage through 400 mesh (37.5μ)	10.9	20.1	37.5	57.6	77.9	11.0	21.4	42.6	67.2	87.7
Specific surface cm^2/g	500	970	1910	3700	6510	520	1000	2020	4020	7160
Specific surface cm^2/cm^3	1330	2570	5050	9790	17230	1370	2640	5330	10630	18950

[a] Mill size: θ 305 x 305 mm
Weight of steel balls: 47.1 kg
Feed: quartz from Radanefors −3.7 + 2.3 mm
Sp gr: 2.646
Specific surface: 18 cm^2/g (44 cm^2/cm^3)
Weight of feed: 5.50 kg

place at a temperature below the freezing point of the gas. Therefore, the adsorbed layer of krypton molecules is many molecules thick.

It is true that with different gases one usually gets different values for the specific surface of the material. These differences are no greater than are often noted in comparing air and water permeability measurements on the same material.[23] The observations of Johnson, Axelson and Piret concerning the effect of degassing temperature on the measured area of a quartz sample also worried me. However, time did not permit study of that problem. It would be an interesting investigation for a physical chemist.

The validity of the B.E.T. method of surface area measurement has been quite well substantiated by the work of Harkins and Jura[24] who devised a so-called absolute method of surface measurement involving no assumption concerning the area occupied by one molecule of gas. They accurately measured the heat evolved upon immersion in water of a crystalline solid that had been previously exposed to saturated water vapor so that the surface energy of the outer film on the solid was the same as that of water. The surface area of the solid is given by the ratio of the heat evolved to the surface energy of water after a small correction has been made for the thickness of the water film. Excellent agreement between the Harkins-Jura and the B.E.T. method has been noted.

Comparisons between gas adsorption and permeability surface area measurements would indeed be interesting.

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A New Theory of Comminution

by F. C. Bond and J. T. Wang

DISCUSSION

H. J. Kamack *(E. I. du Pont de Nemours & Co., Inc., Wilmington, Del.)*—Rittinger's law usually is stated to the following effect: "The work (or energy) consumed in particle size reduction is proportional to the new surface area produced." The law has been stated substantially in this way by Taggart,[20] Berry,[21] Dalla Valle,[22] Coghill and DeVaney,[23] Richards and Locke,[24] Gross,[25] and many others, and, according to Gaudin,[26] was originally expressed by Rittinger in the same form. Consequently there can be little doubt that this is the understanding of the law among most workers in the field of particle size reduction. Bond and Wang, however, express the law in the form "the *useful work accomplished* . . . ." (italics theirs). The distinction is critical, for in the form used by Bond and Wang the law becomes, as they themselves remark "merely a more or less arbitrary definition of useful work," while in its usual sense the law expresses a physical hypothesis which has been verified experimentally within certain limitations.

Considering the way the law has been used, it might be stated more explicitly as follows: "In a given machine operating under a given set of constant operating conditions, the work consumed in the particle size reduction of a given material is proportional to the new surface area produced." Or, as Coghill and deVaney have said,[27] "the (Rittinger) law holds only when the tests being compared are made under analogous conditions." There are occasions when the law transcends these limitations; for example, the surface area produced per unit energy consumption for a given material in a ball mill does not vary much over a fairly wide range of operating conditions. But by and large, the surface area production per unit energy consumption will vary with the operating conditions, the type of machine, and the material. The essence of Rittinger's law is that the surface area production per unit energy consumption is independent of the particle size, and this has been verified experimentally by numerous workers for numerous materials, within certain limits. An important limitation is that when one grinds to very small particle sizes, agglomerative forces may tend to interfere with size reduction so that the surface area increases less rapidly than Rittinger's law would predict. Within such limitations, Rittinger's law can be regarded as empirically established.

The law has, however, certain theoretical implications, and it seems to be chiefly against these that Bond and Wang direct their criticism. Solids are believed to possess surface energy which is proportional to surface area. Thus, Rittinger's law implies a proportionality between surface energy produced and mechanical energy expended (for a particular material in a particular machine). It does not imply that all or most of the mechanical energy is transformed into surface energy; in fact it is known that most of the mechanical energy is transformed into heat. Bond and Wang assert that most of this heat arises from the damping of elastic vibrations of stressed particles. This may possibly be true for crushing (with which they are chiefly concerned), although in grinding it is probable that much of the heat arises from friction between particles. However, the fact that the surface energy is small compared to the heat energy does not invalidate Rittinger's law, which implies merely that they are proportional.

The authors also criticize Rittinger's law on the grounds that "this theory cannot be justified mathematically, since work is the product of force times distance, and the distance factor is ignored," and "energy input must be the product of force times distance, and the Rittinger theory completely ignores large variations in the distance (strain dimension or deformation) throughout which a force must act to produce breakage of different materials." However, the quantities force and distance as such are irrelevant to Rittinger's law, which considers energy input as an integral quantity. The fact that different materials require different forces and strains (hence, different amounts of energy) to break them is incontrovertible but again is irrelevant to Rittinger's law, which, as mentioned before, applies only to a single material. Even in theory, it would not be expected that the surface area produced per unit energy consumption would be the same for different materials, because the surface energy per unit area is presumably specific to each material.

Bond and Wang advance another argument of a

Table VI. Grinding Test of Sand

Grinding Time, Min	Specific Surface Average Particle Diam, Microns[a]		New Surface Area, Sq M		80 Pct Size, Microns[b]	
	35/48 mesh sand	100/150 mesh sand	35/48 mesh sand	100/150 mesh sand	35/48 mesh sand	100/150 mesh sand
0	349	124	0	0	360	130
60	7.1	8.5	840	664	53	68
120	4.07	4.5	1460	1291		

[a] Measured with a Fisher Sub Sieve Sizer with results corrected for *slip flow*. Feed sizes determined from sieve analyses.
[b] From plot of sieve analysis.

different nature, saying that "The Rittinger theory . . . . has been criticized as assigning too much energy to reduction of the very fine particles, while requiring the use of a more or less arbitrary grind limit." This criticism should properly be applied to the method employed for determining surface areas; i.e., calculation from size distributions, using the Gaudin-Schuhmann law for extrapolation. It is a consequence of this law that, if the slope of the cumulative distribution curve is less than 1 (as is usually the case for comminuted materials), then the total surface area on any size grade increases without limit as the particle size of the grade decreases, so that, unless some arbitrary limiting particle size is assumed, the calculated surface area of the powder is infinite. This would indicate that the Gaudin-Schuhmann relationship is physically untenable, even though it affords a fairly good approximation to part of a size distribution. Equally good or better approximations can be obtained with other methods of plotting, notably the method of plotting on log-normal (also called log-probability) paper.[28] Such paper is available commercially.[29] A straight-line extrapolation on this paper will always yield a finite surface area for the powder, and areas so calculated usually agree fairly well with direct measurement of surface area (e.g., by the permeability method).

Strain-Energy Theory: According to this hypothesis, the work consumed in size reduction of a particular material is proportional to $(n+2)(n-1)/n$, where n is the reduction ratio.* This expression is approximately equal to n (the difference never exceeds 12.5 pct of n when n is greater than 1.66). When the cumulative size distributions of feed and product are nearly parallel, Bond and Wang define n as F/P, where F and P are the sizes which pass 80 pct of the feed and product, respectively. When the feed is closely sized, Bond and Wang describe a method for calculating an average value of n. In either case, it can be shown that n represents approximately the ratio D_f/D_p, where D denotes specific surface average particle size and the subscripts f and p refer to feed and product, respectively. D (in cm) is defined by the relation

$$D = 6/(SA)(Sp)$$

where SA is the specific surface area of the material in sq cm/g and Sp is its specific gravity.

According to the strain-energy theory, then, the work consumed in size reduction of a particular material is proportional to D_f/D_p, while according to Rittinger's law it is proportioned to

$$\frac{1}{D_p} - \frac{1}{D_f}$$

Consequently, if one starts with a given feed size, and comminutes until the reduction ratio is fairly large, both Rittinger's law and the strain-energy law predict that the energy consumption will be inversely proportional to the specific surface average particle size of the product. (Also, if the slope of the cumulative size distribution curve of the product is assumed to remain constant as comminution progresses, then the energy consumption will be inversely proportional to F). This is why in Fig. 2 of the paper the curves for the *modified* Rittinger method and the strain-energy method are parallel, both for *crushing* and for *grinding*.

On the other hand, if one were to start with two different feed sizes, of the same material, and apply the same amount of energy to each, sufficient to produce a fairly large reduction ratio, the Rittinger law would predict that the product sizes would be nearly equal, whereas the strain-energy theory would predict that the product sizes would be proportional to the feed sizes. This suggests a simple crucial test of the theories, which we have made as follows. Two closely-sized grades of sand, one 35 to 48 mesh, the other 100 to 150 mesh, were ground dry for the same length of time under the same conditions in a 12 in. diam x 6 in. ball mill (speed, 51 rpm; charge, 47 lb of ¾ in. steel balls, 5.9 lb of sand). The ratio of the mean feed sizes, therefore, was 2.8 to 1.

The measurements of surface area and specific surface average particle size are in approximate agreement with Rittinger's law but in violent disagreement with the strain-energy theory. Also, the 80 pct sizes of the products after 1 hr are 53 and 68 microns, again in violent disagreement with the strain-energy theory. (The fact that the coarser feed gave a slightly finer product we attribute to a small difference in grindability of the two grades.)

Evidence Adduced for the Strain-Energy Theory: Bond and Wang offer Fig. 2 of their paper as evidence of the correctness of the strain-energy theory. This graph is a log-log plot, with energy consumption per unit weight of material as ordinate and with the expression $X = F^{1/2}/P^{3/2}$ as abscissa. Empirical data yield a set of curves with a slope of 0.5. The strain-energy theory may be written

$$h = kF/P$$

where h is the energy consumption and k a constant characteristic of the material. This may be written as

$$\log h = \log k + \frac{2}{3}\log F + \frac{2}{3}\log X$$

which shows that on the graph the strain-energy theory is represented by a straight line with a slope of 2/3. This disagrees with the empirically determined slope of ½. That the strain-energy curve for crushing lies in the same region as the empirical data is fortuitous because, as the equation above shows, the actual position of the curve on the graph may be shifted at will by varying the feed size, F. This is also indicated in the figure itself, because the curves labeled "crushing" and "grinding" represent two arbitrary choices of feed size.

The curves for the Rittinger law may also be freely translated on the graph by varying the feed size. For fairly large reduction ratios, the Rittinger curves nearly parallel the empirical curves. The reason for this is that the surface areas were calculated upon the Gaudin-Schuhmann law, and it can be shown from this that D_p is approximately proportional to P^m. Thus, Rittinger's law:

$$h = k\left(\frac{1}{D_p} - \frac{1}{D_f}\right)$$

becomes, for a fairly large reduction ratio

$$h = k'/P^m$$

or

$$\log h = \log k' - \frac{1}{3}m\log F + \frac{2}{3}m\log X$$

so the slope is $\frac{2}{3}m$. Bond and Wang use a value of m

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[*] The notation of Bond and Wang is followed throughout this discussion.

$= 0.7$, so the slope is 0.47, or nearly the same as the empirical slope. Thus one could conclude that the Rittinger law agrees better than the strain-energy theory with the empirical data presented by Bond and Wang.

The *modified* Rittinger law, which Bond and Wang also plot, consists in replacing D_p and D_f with P and F in Rittinger's law. Thus, for a fairly large reduction ratio, the modified Rittinger law can be expressed as

$$h = k/P$$

or

$$\log h = \log k - \frac{1}{3} \log F + \frac{2}{3} \log X$$

This curve, therefore, parallels the strain-energy curve, as has already been noted.

Conclusion

The arguments which Bond and Wang present against the Rittinger theory do not stand up under critical examination. The "strain-energy" theory which they propose instead is not supported by the empirical data they present and is disproved (for grinding) by an experiment which shows that for sand the energy required for grinding is not necessarily proportional to the reduction ratio.

R. G. Wuerker *(University of Illinois, Urbana)*—The strain-energy theory, as used by the authors, is based on the use of the modulus of resilience as a criterion for the failure of a material. The quantity of strain energy stored per unit volume, as expressed in the term $S^2a^3/2E$, step 1, has been proposed as a basis for determining failure as far back as 1885 by Beltrami.[31] Its application to failure of a mine roof has recently been discussed by Philipps.[32] Its use in the study of comminution, steps 2 and 3 in the authors' proposal, is therefore well founded and full of promise.

If the modulus of resilience should be correlative with quantities used in comminution, like impact crushing strength, energy per new surface created, etc., much time and expense could be saved. For the area under the elastic part of the ordinary stress-strain diagram is equal to the modulus of resilience and could be easily graphed or computed.

However, data listed in the literature do not allow such a correlation attempt yet. One never finds crushing and grinding data of rocks, of which also fundamental properties like modulus of elasticity and compressive strength, or vice versa are known. It would be indeed a worth-while research project to make the necessary measurements and correlations of rocks considered typical for their behavior in crushing and grinding.

The authors have stressed the need for more accurate determinations of the modulus of elasticity. Besides accuracy, an agreement is needed, which modulus we intend to use. For with brittle materials like rocks, E is not a unique quantity, not a constant, as in the case of ductile metals. The stress-strain curve has very often no straight line portion at all. In the case of concrete it is customary to work 1—with the tangent modulus through the point of origin, 2—with the tangent modulus for a given stress, and 3—with the secant modulus at the ultimate limit stress.

These three moduli are arrived at by static testing. But recently sonic testing has come to increased use. E determined by this latter method is always the initial tangent modulus through the point of origin and is therefore a higher value than many moduli of elasticity listed in the literature which have been obtained by the static method. In a recent publication of the U. S. Bureau of Mines[33] are some examples (group numbers 29.2 and 30.1) where E obtained by the sonic method is three or four times as high as the values obtained by the conventional static method. If one is going to use indiscriminately any modulus of elasticity, without stating how it has been obtained, not much accuracy can be expected.

It is suggested to reach an agreement first, which E should be used in the case of comminution computations.

F. C. Bond (authors' reply)—The distinction between total work input and useful work accomplished in the statement of the first form of the Rittinger theory is important, since it is obvious that in a machine whose mechanical efficiency is different at different product sizes the work consumed in particle size reduction is not proportional to the new surface area produced; unless it should happen that the true relationship between useful work accomplished and surface area produced should compensate for the difference in mechanical efficiency. In a given machine operating under a given set of constant operating conditions the absolute mechanical efficiency is not necessarily constant at different feed and product sizes.

The power required to rotate a ball or rod mill when the feed is cut off is approximately the same as that required when grinding, and it is difficult to imagine that when 99 pct of the feed is cut off the relationship between energy input and surface area production remains constant.

I seriously question the statement that "Within such limitations, Rittinger's law can be regarded as empirically established." Most of the published experimental confirmations of the Rittinger theory start with an unnatural feed of particles of constant size, and often of uniform size, from which the fines have been removed. A valid confirmation of the theory requires that a feed be used with the same normal plotted size distribution slope as the product; with a scalped feed the relative amounts of work done on the different size fractions of the product are entirely disproportionate. Tests made with a scalped feed may appear to confirm the Rittinger theory while concealing the true theory from discovery. To claim that a scalped feed makes no difference is to assume a priori the proposition to be proved. In considering confirmatory tests made on quartz it should also be remembered that grindability tests on pure crystallized quartz[34] do not parallel those made on most ores, the quartz being relatively harder to grind at fine mesh sizes than average materials. There is also important experimental evidence that the theory does not hold.[35]

The second form of the theory involves the concept of surface energy, and two schools of thought exist on this subject.

One believes that all of the useful input energy is transformed into surface energy. It is exemplified by the statement of Professor Gaudin that "—the efficiency of a comminution operation is the ratio of the surface energy produced to the kinetic energy expended." If this is true, we can hope for some wonderfully welcome decreases in the energy required to crush and grind, since present efficiencies under this definition are around 1 pct or less.

The other school believes that the surface energy produced is a constant, though probably small, proportion of the energy required. However, it has never been clearly explained just why the ratio should be constant, except that the logic of the Rittinger theory demands it. It appears much more accurate to depend upon the total energy input, rather than upon the constancy of the small proportion of it represented by the surface energy.

If the energy required to form a unit of new surface area of different materials is directly proportional to the unit surface energies of these materials, and the energy required for different size reduction is directly proportional to the new surface produced, then the Rittinger theory applies to all materials; and if the energy required to break is the product of average force times the distance of deformation it appears that the energy required cannot be independent of the amount of deformation, as the Rittinger theory requires.

The work of Carl Zappfe[36, 37] supports the existence of micelles of about colloidal size in metals, and it would seem logical that they exist in rocks as well, thus postulating the existence of a grind limit to the Gaudin-Schuhmann distribution line. It is noteworthy that the calculated total surface areas of ground products by any of the "absolute" methods are similar to those obtained by the G-S line and a grind limit of 0.7 micron, with a suitable correction for the surface areas of irregular particles.

If the G-S line is straight, the log-probability plot of the line is curved. The product of an open circuit grind product may sometimes yield a straight log-probability line and a curved G-S line, but this probably results from slabbing of the larger particles and tramp oversize. The G-S line of a closed circuit product is usually straighter than the log-probability line, and the G-S plot is more generally used.

Since the paper was written I have appreciated that some of the same objections apply to the strain-energy theory that apply to Rittinger and Kick: namely, all are theoretically explained by the regular breakage of cubes, which is not similar to actual crushing and grinding of irregular pieces of rock; and all deal primarily with increments of work from the feed size, rather than with the total work from theoretically infinite particles of feed.

In Mr. Kamack's grinding test the confirmation of the Rittinger theory was made in the usual manner previously criticized; an unnatural feed of sized particles was used. The fact that the coarser sand feed gave a finer product size and more surface area after a constant work input shows that considerable difference exists between the grindabilities of the two grades of sand. One calculation indicates that the 100/150 mesh sand is 2.66 times as hard to grind in the same size range as the 35/48. It is obvious that no valid comparison of the theories can be made between two such different materials.

Fig. 2 was prepared to illustrate objectively the comparison of the different theories, and not as evidence of the correctness of the strain-energy theory. The difference in slope between the empirical lines and the strain-energy lines, and the lateral displacement of all the theoretical lines with changes in the feed size, are all shown in the figure, and it is true that one might conclude from Fig. 2 that the Rittinger lines may agree as well with the empirical data as the strain-energy lines. We did not attempt to prove the correctness of the strain-energy theory in this paper, but presented it as fairly as possible as a logical alternative to the Rittinger and Kick Theories. It is a development of the laws of mechanics dealing with the breakage of physical structures, and as such is worthy of consideration. None of the theories agrees too well with data from actual plant operation.

The Rittinger theory superficially appears logical, but the real test of a theory lies in its practical application. After many years of intensive study and effort it has still not been correlated satisfactorily with operating results. It remains practically valueless in the solution of actual crushing and grinding problems, which must still be solved by empirical comparisons within narrow size ranges. A correct theory, which will permit direct comparisons of efficiency between different materials in different machines over all size ranges from easily obtained data, is needed. The strain-energy theory resulted from a logical attempt to fill this need, and we believe it is a step in the right direction. Development of the correct theory will permit numerical evaluation of the many diverse factors which affect crushing and grinding, and result in more accurate predictions regarding new installations, leading to the development of more efficient comminution methods. General acceptance of the Rittinger theory may have retarded discovery of the correct theory, and retained the industry in the empirical darkness in which we are all now groping. The light certainly exists, and the strain-energy theory represents one attempt to find it.

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Comparative Results with Galena and Ferrosilicon at Mascot

by D. B. Grove, R. B. Brackin and J. H. Polhemus

DISCUSSION

P. L. Jones *(Sink and Float Corp., New York)*—The comparisons between galena and ferrosilicon medium should be applied only to the specific sink-float process used at Mascot since no evidence is presented that allows the comparison to be extended generally to galena medium as now used in other forms of sink-float.

It is stated that flotation recovery of galena medium was tried but was unsuccessful and therefore the elaborate gravity system of medium recovery was applied. The expense of the gravity system at Mascot was one of the reasons for changing to ferrosilicon. However two important sink-float plants treating a combined total of over 13,000 tons of ore per day are using flotation to clean galena medium most successfully and are obtaining medium recoveries comparable with those reported in the paper for ferrosilicon.

Slightly increased zinc recovery is claimed due to the change-over but it is to be noted that at the time of the change a decreased density differential was obtained in the separator. With galena a differential

Table XII. Based on Results in Table II in the Paper

Product	Weight, Pct	Zn Assay, Pct	Zn Distribution, Pct
Sink	21.6	10.33	90.5
Float	78.4	0.297	9.5
Cone Feed	100.0	2.46	100.0

Table XIII. Theoretical Results Based on Float Elimination Retaining Actual Float Elimination But Using Zinc Distribution Figures Obtained in the Tests.

Product	Weight, Pct	Zn Assay, Pct	Zn Distribution, Pct
Sink	21.6	10.64	93.3
Float	78.4	0.21	6.7
Cone Feed	100.0	2.46	100.0

Table XIV. Alternatively Theoretical Results Based on Actual Zinc Distribution, i.e. Retaining Actual Zinc Distribution but Using Weight Distribution Figures Obtained in Tests.

Product	Weight, Pct	Zn Assay, Pct	Zn Distribution, Pct
Sink	16.0	13.94	90.5
Float	84.0	0.28	9.5
Cone Feed	100.0	2.46	100.0

Table XV. Metallurgical Results for August 1951.

	1947, with Galena	First 6 Months 1950, with FeSi	August 1951
Tailing Assay, Pct Zn	0.310	0.297	0.264
Concentrate Assay, Pct Zn	12.08	10.33	12.06
Heavy Media Separation Recovery, Pct	89.38	90.22	90.99

density of 0.15 was used.[1] Although not so explained in the paper, this decreased differential is presumably due to the fact the new galena medium was —10 mesh whereas the new ferrosilicon medium is —65 mesh. The effect of a high density differential is to decrease mineral recovery as suggested by work done at the Central Mill of the Eagle-Picher Company at Picher, Oklahoma.[2] Marginal sink particles are apparently prevented by the high density differential from reaching the sink product outlet.

The figures in Table XII were taken from Table II and show the good metallurgical work done.

To compare with these results, heavy liquid separation tests were made on composite samples during two periods separated by eleven months and gave remarkably similar results, suggesting there is but little variation in average ore. These are shown in Fig. 5 in the paper. Using these results it is possible to compare theoretically perfect density separation with actual as given in Table XIII.

These results illustrate that however good the results seem they could be bettered. Thus if a 78.4 pct float elimination as actually obtained was acceptable, with theoretical separation a float tailing of 0.21 pct Zn should have been obtained, Table XIII; whereas actually 0.297 pct Zn was obtained, Table XII. Looked at another way, if a 90.5 pct Zn recovery as actually obtained was acceptable, with theoretical separation an 84 pct elimination of float should have been obtained, Table XIV, but actually 78.4 pct elimination was obtained, Table XII.

These results illustrate that looked at either way the sink contains considerable float and the float contains considerable sink. It seems that this can be expected in any sink-float process that uses a density differential as large as that used at Mascot even after the changeover.

D. B. Grove, R. B. Brackin, and **J. H. Polhemus** (authors' reply)—The purpose of the paper was to show the major improvements resulting from the conversion to ferrosilicon at Mascot. No attempt was made to extend the comparison between galena and ferrosilicon to other forms of sink-float. However, this could readily be done to the extent that wherever ferrosilicon could be substituted for galena, medium costs would be reduced because of the lower cost of ferrosilicon, even if no reduction in medium consumption were effected.

The flotation system of cleaning medium at Mascot was not as effective as the gravity method or it never would have been abandoned. It is a fundamental fact that galena used in a medium system is exposed to oxidation, which complicates its recovery by flotation. The ferrosilicon loss at Mascot is 0.15 lb per ton of ore milled. On the same ore the galena loss was 0.80 lb per ton milled. There can be no question as to the reliability of this figure, since the Mascot ore is lead free, making it much easier to determine the exact medium consumption than in a plant where the galena is being obtained from the ore milled. It would be of interest to see the exact figures on medium consumption from the two large plants using flotation to clean galena and obtaining medium recoveries comparable to the 99.9 pct recovery of ferrosilicon made at Mascot. Even if this recovery could be equalled with galena, the unit cost of the latter is approximately double that of ferrosilicon, which gives a much higher medium cost.

The expense of the gravity cleaning system at Mascot was but one reason for changing to ferrosilicon. The primary reason was the high cost of galena and the difficulties involved in handling this soft, friable medium which is inherently more difficult to recover than a magnetic medium.

The higher differential in the cone with galena was due to the necessity of using 10 mesh galena. Finer sizes comparable to the 65 mesh ferrosilicon now in use were tried but ruled out because of exorbitant losses of fine galena. Generally speaking, the finer the galena used, the higher will be the medium loss. This factor had to be balanced against any reduction in tailing assay that might have been obtained by operating with a lower differential. Three years operating experience with ferrosilicon have shown that the best results on Mascot ore are made with a differential of 0.08 to 0.10 in the cone. It is our belief that the question of whether or not differential is required in a sink-float operation and the extent of this differential can be determined only by actual operation on the individual ore, and that no generalizations on this question are possible.

The plant is currently treating +¼ in. ore, a much finer feed than the galena circuit was able to handle, and the results are the best that have been obtained in the history of the operation. The metallurgical results for August 1951 are shown in Table XV beside the comparable figures given in the paper, indicating that the further improvements in metallurgy predicted in the paper are being obtained.

It is recognized that the results being obtained could be bettered, as Mr. Jones points out. This is true in any ore dressing process. However, the above tabulation shows that marked improvement has been made since the paper was written, and this improvement is in no way due to any change in differential. Rather, it is due to minor changes in operating technique and increased use of recording instruments to keep the operators informed of tonnage rate, medium gravity and medium volume. Other refinements of this nature may bring about further improvements.

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[1] Differential Density Separation at Mascot, Tennessee, Engineering and Mining Journal (July 1940).

[2] U.S. Bureau of Mines, R. I. No. 4511, May, 1949.

The Probability Theory of Wet Ball Milling and Its Application

by Elliott J. Roberts

DISCUSSION

F. C. Bond *(Allis-Chalmers Mfg. Corp., Milwaukee)*—This paper considers comminution as a first order process, with the reduction rate depending directly upon the amount of oversize material present. The data show that other factors should be taken into account, and it is possible that in time these may be evaluated as simultaneous or consecutive reactions. Development of the theory of comminution has been retarded for many years by the assumption that surface area measurements constitute the sine qua non of the work done in crushing and grinding, and it is encouraging to note the belated growth of other ideas.

In the Abstract the term "net power" should be changed to "net energy." Throughout the paper the term "hp per ton" should be changed to "hp hrs per ton", or "hp hr t."

The term "Probability Theory" in the title does not seem appropriate, since it is not clear how the probability theory is used in developing the ideas in the paper.

There seems to be a contradiction between the large calculated advantages of closed circuit operation and the statement following that the closed circuit test results showed no significant change in grinding behavior, when compared with the batch grind curves.

Tables I and II show that between 75 pct and 50 pct solids the energy input required decreases with increasing moisture content and may indicate the advisability of grinding at higher dilutions in certain cases.

The calculation of the hp-hr per ton factor indicates an input in the laboratory mill of only 7.32 gross hp per ton of balls; this casts some doubt upon the accuracy of the factor used, since the power input in commercial mills at 80 pct critical speed is customarily much higher.

The tests show that within fairly wide limits the amount of ore in the laboratory mill may be varied and a product of constant fineness obtained, provided that the grinding time is varied in the same proportion. This has often been assumed, and confirmation by actual testing is of value.

The C_{avg} corrections for differences between the plant and laboratory size distributions do not seem very satisfactory, since in many cases the plant/laboratory ratio is farther from unity after correction than before.

The following equation has been derived from the data in Table VI:

$$\text{Relative Energy Input} = \frac{(\text{log new ball diam in in.} + 0.410)}{0.844}$$

from which the relative energy inputs for balls of different sizes can be calculated and compared. The relative energy input is unity for balls of 2.715 in. diam. The equation indicates that the work accomplished by a ton of grinding balls per unit of energy input is roughly proportional to the square root of the total ball surface area; provided, of course, that the balls are sufficiently large to break the material. The data in support of this statement are admittedly meager, but are fairly consistent when plotted.

The relative grindability values listed in Table VI for 200 mesh multiplied by 4/5 apparently correspond approximately to the A-C grindability at 200 mesh.[5]

It would seem that for open circuit tests comparable accuracy could be obtained much more simply by the old method[5] of plotting the test grind, extending the mesh grinds to the left of zero time if necessary, and determining from the plot the equivalent time required to grind from the plant feed size to the plant product size, using the average of several mesh sizes. The energy input value of one time interval could be determined by tests on materials of known grinding resistance, and this multiplied by the interval required should give the desired energy input value. The relative grindabilities would be the relative time intervals required for a specified feed and product size.

When the plotted mesh size lines of a homogeneous material are extended to the left beyond zero time they meet at one point at zero pct passing. The horizontal distance of this point from zero time indicates the equivalent energy input required to prepare the mill feed.

The author's results show that the closed circuit grinding tests give about the same K values as open circuit tests, from which he concludes that open circuit tests are satisfactory in many cases. The value of the closed circuit test is its ability accurately to predict energy requirements in closed circuit grinding for both homogeneous and heterogeneous materials. If the material is homogeneous, the open circuit test gives satisfactory results; but if the material contains appreciable fractions of hard and soft grinding ore, the open circuit tests will not be accurate because of the accumulation of hard grinding material in the circulating load. Since in most cases it is not possible to determine a priori whether the material contains hard and soft fractions, the closed circuit tests are preferable and more reliable.

B. S. Crocker *(Lake Shore Mines, Ontario)*—Dr. Roberts probability theory of grinding is very similar to our log pct reduced vs. log tonnage method of plotting and evaluating grinding tests at Lake Shore. However, although we both seem to start at the same point we finish with different end results.

Shortly after publishing our grinding paper (referred to by Dr. Roberts) in 1939, we did pursue the subject of the "constant pct reduction in the pct +28 micron material for each constant interval of time. We ran innumerable tonnage tests on the plant ball mills, rod mills, tube mills with 1¼ and ¾ balls, and lastly pebble mills, with tonnage variations from 180 tons per day to 950 tons per day. We found that when we plotted the log of the tonnage against the log of the pct reduced of any reliable mesh, we had a straight line up until 90 pct of the mesh is reduced. We have also tested this in our 12-in. laboratory mill with the same results. We have used this method of evaluating grinds for the past 8 years and developed the recent four stage pebble plant on this basis.

By pct reduced we mean the percentage of any given mesh that is reduced in one pass through a mill at a given tonnage (or time). For example, if the feed to a rod mill is 90 pct +35 mesh and the discharge at 500 tons per day is 54 pct +35, the pct reduced is

$$\frac{90 - 54}{90} = 40 \text{ pct.}$$

If the feed had been 80 pct +35 the discharge would have been 48 pct +35 or pct reduced

$$\frac{80 - 48}{80} = 40 \text{ pct}$$

as long as the tonnage remained constant at 500 tons per day. Thus we can easily correct for normal variation of mill feeds. This log — log relationship derived from the tonnage tests of all our operating mills has proved of tremendous help in checking laboratory work and in designing alternate layouts or new plants.

The difference between the log — log and the semilog plot is only shown up when the extremes in tonnages are plotted.

When the relationship between the pct reduced and the tonnage was first investigated, we used semilog paper and apparently had established a straight line relationship between 500 and 800 tons per day on the ball mill and rod mills. However when the points below 500 (down to 190) and above 800 (up to 930) were plotted on semilog paper the two extremes were definitely not on the straight line on semilog paper but all points were on a straight line on log-log paper. We therefore, changed to the log-log plots and subsequent work has confirmed its reliability. Table VIII and Fig. 5 shows a recent tonnage test on our rod mill. The pct reduced for all meshes are plotted in this case. Usually only one or two key meshes are worked on.

We agree with Dr. Roberts that quick reliable work can be done in small laboratory mills. We have used a 12 in. diam by 7½-in. long laboratory mill since 1934 to prove all plant changes done on —8 mesh material. This includes ¾-in. ball testing and the recent pebble grinding with screened ore.

E. J. Roberts (author's reply)—It would be highly desirable if Mr. Crocker could find the time to expand his most welcome discussion into a full length paper. The writer has not had opportunity to make an exhaustive comparison of the two methods of handling grinding results; but, applying Crocker's method to the original data of the paper under discussion, it is found that only rarely do straight lines result by log-log plotting. They are straightest on the coarser meshes of another Kirkland lake ore (Ore C), but on Ore A all of the plots are decidedly concave toward the low tonnage—low percent reduced corner of the paper.

It may be that the use of both methods of plotting would prove fruitful in getting the most out of a set of data. One objection the writer has to the log-log method is that the sensitivity of the plotting is greatest where the screening accuracy is the least and vice versa. To illustrate; at high tonnages, the percent reduction is low; and the accuracy of this figure is low because it is obtained by taking the difference between two relatively large numbers. Yet on log-log paper, the difference between 9 and 10 pct reduction is over ¼ in displacement on single cycle paper. On the other hand at low tonnages and high percent reductions, the error as a result of screening may well be only about 50 pct of the other error; but the corresponding displacement even for the same error would only be about 1/32 in. (between 89 and 90 pct).

The above may not be the whole story because the effect of other errors should also be considered.

Mr. Bond is in error on both points of his second paragraph. The wording of the abstract describes a rate process and therefore power and not energy is the proper word. For the same reason, through the paper whenever hp/ton is mentioned, hp/ton is meant.

The writer was reluctant to use the term probability theory because of possible confusion with error curves etc. However, the basic thought of the writer's probability theory is the same as that of the first one of the probability theorems mentioned by Daniels[8] "The probability of an event depends on the number of possibilities."

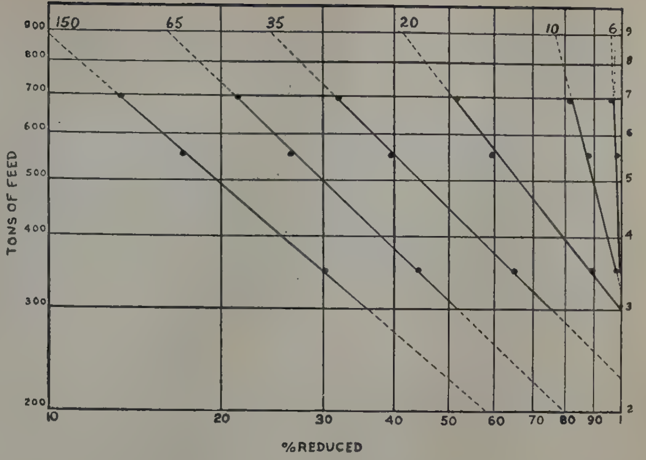

Fig. 5—Tonnage tests, 2½-in. rods, No. 5 rod mill, Nov. 6-29, 1950.

The statement that the closed circuit results showed no significant change in grinding behavior was meant to convey the thought that eq 4 still held and therefore there was no build up of hard particles.

Table I clearly shows that the higher the moisture content, the coarser the product of a 2-min. grind; and, therefore, the energy input required to obtain a given product must *increase* with moisture.

The reason for the low figure of 7.32 hp/ton of balls in the laboratory mill is the small diameter of the mill. Other things being equal, the hp per ton of balls increases as the square root of the diameter, see ref 4.

The C_{avg} corrections may well make the plant laboratory ratio farther from unity after correction than before. There is nothing sacred about unity here except for 2½-in. balls. The ratios for other ball sizes will naturally have to be different from unity.

In deriving any relationships regarding the effect of ball size on power from Table VI, the last two ratios (0.76 and 0.60) must not be used, because the data are heterogeneous with respect to the rest of the data. This is because in these last two cases the laboratory feed was 20 mesh instead of 4 mesh. This makes a significant change in the laboratory K's. The rest of the data correlate reasonably well on the formula:

hp ratio plant/lab = 0.73 (makeup ball diam) 1/3, eq 15

which is the relationship we use. However, as stated in the paper the data are neither sufficiently extensive nor accurate to permit any firm deductions on this point.

Table VIII. Screen Analyses—Tyler Standard Screens, 2½ in. Rods, Tonnage Tests, No. 5 Rod Mill

Tons Per Day Tonnage Range No. of Tests			352 Tons 337 to 366 8			560 Tons 530 to 590 2			697 Tons 690 to 706 4		
Openings In.	Openings Mm	Mesh	Feed, Cum. Pct	Disch., Cum. Pct.	Pct Reduced	Feed, Cum. Pct	Disch., Cum. Pct	Pct Reduced	Feed, Cum. Pct	Disch., Cum. Pct	Pct Reduced
0.371	9.423		0.7						0.7		
0.263	6.680	3	15.0			15.8		100.0	17.0		100.0
0.185	4.699	4	35.2		100.0	35.5	0.2	99.4	37.2	0.3	99.1
0.131	3.327	6	49.9	0.1	99.8	51.2	0.9	98.0	52.3	1.7	96.7
0.093	2.362	8	59.9	0.4	99.4	61.5	3.1	94.8	61.7	5.6	91.0
0.065	1.651	10	68.2	0.9	98.5	69.7	8.2	88.2	70.6	13.2	81.2
0.046	1.168	14	74.9	3.3	95.7	76.0	20.1	73.5	76.9	27.1	64.8
0.0328	0.833	20	79.4	8.6	89.2	80.0	32.9	58.8	80.9	39.5	51.2
0.0232	0.589	28	82.5	17.6	78.6	82.8	43.7	47.2	83.9	49.5	41.0
0.0164	0.417	35	85.5	29.9	65.0	85.7	52.3	39.0	86.4	58.6	32.2
0.0116	0.295	48	87.5	40.4	53.8	87.5	60.8	30.5	88.1	64.9	26.4
0.0082	0.208	65	89.3	50.0	44.0	89.2	66.0	26.0	89.8	70.4	21.6
0.0058	0.147	100	90.8	57.8	36.4	90.7	72.0	20.6	91.1	74.9	17.7
0.0041	0.104	150	92.1	64.1	30.4	92.0	76.2	17.2	91.9	79.6	13.4
0.0029	0.074	—150	100.0	100.0		100.0	100.0		100.0	100.0	

If eq 15 is correct, the relative energy input is unity for a 2.56 in. ball, and, for what it may be worth, the work accomplished by a ton of balls per unit of energy input is proportional to the cube root of the ball surface area.

For open circuit estimates, we do essentially what Mr. Bond suggests, if the writer understands him correctly.

With regard to Mr. Bond's last paragraph, the writer merely stated that the batch grind data was used for obtaining a relative grindability series. In practice these figures are used for rough estimates. For more precise estimates, a closed circuit test is made at the particular mesh desired. The greatest danger in using these batch figures for closed circuit estimates comes from the effect of classification on minerals of different densities. Pyrite in the overflow of a classifier is much finer than quartz and as a result the average feed to the ball mill has a different ratio of pyrite to quartz than that of the raw ore and the grindability may be different. Also the last column of Table VII will no longer apply.

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[5] F. C. Bond: Standard Grindability Tests Tabulated. *Trans.* AIME (1949) 183, p. 313. Mining Technology (July 1947). TP 2180.

[8] *Mathematical Preparation for Physical Chemistry,* Farrington Daniels, McGraw Hill Book Co. Inc. (1928).

Progress Report on Grinding at Tennessee Copper Co.

by J. F. Myers and F. M. Lewis

DISCUSSION

L. E. Djingheuzian *(Canadian Dept. of Mines and Technical Surveys, Ottawa)*—In their Summary the authors say: "Reconciling the grinding efficiency with good metallurgy is still a problem."

In the discussion of the first paper[3] in his reply to W. I. Garms, Mr. Myers states:

"Our grinding process with smooth 1-in. balls has reduced by nearly one half the metallic losses in the fine micron sizes of the tailing. This is simply because less of the fine micron sizes are produced. Since the + 65 mesh size is the same as formerly, a higher percentage of the intermediate sizes are developed. These sizes have the *highest floatability,* require the *least reagents,* and use *less floating time.*

"These factors contribute so heavily to the overall economies that dropping our power grinding gain from 28 pct back to 19 pct is a small detail. However, we feel that this is only a momentary situation and that eventually the best features of the grinding and flotation processes can be brought together, which is as it should be."

Italics are mine.

The above statements, to me, appear to be the answer to the opening statement in the Summary.

Denoting the costs at different power grinding gains as:

	Power Grinding Gain, 28 Pct	Power Grinding Gain, 19 Pct
Cost of grinding	G	G_1
Cost of flotation	F	F_1
Value of metallic losses	T	T_1

where $G_1 > G$, $F_1 < F$, and $T_1 < T$, we have:

$$G_1 + F_1 + T_1 < G + F + T.$$

Since the authors accept the idea that "grinding in flotation plants becomes part of the 'conditioning' of the feed to flotation",[4] i.e., that in flotation the ball mill is primarily a conditioning machine, it can be postulated that Tennessee Copper grinding at cost G_1 is more efficient than grinding at lower cost G. This can be directly inferred from the Conclusion of the paper. Mr. Myers also emphasizes this at the end of his reply to Mr. Garms: "that grinding is for the purpose of preparing flotation feed and not grinding per se."

This, to me, in the final analysis means that when the efficiency of grinding is weighted against the conditioning factor, the former becomes a function of efficient conditioning, hence, within the system in which proper conditioning is the dominant factor, the *best* grinding efficiency is provided by grinding which will contribute towards the optimum conditioning. This brings us again to the statement: "that if every grinding unit were considered as a conditioner for each following step, efficient grinding plants would become much easier to design."[4] In other words, grinding equipment should be balanced against the flotation equipment and against chemical reactions taking place in the system.

F. C. Bond *(Allis-Chalmers Mfg. Co., Milwaukee)*—The authors' discussion of the probable ball motion in a slow speed high dilution mill is very interesting. When the 1-in. balls have worn down to about one fourth of their original weight they apparently first develop a flat surface; as wear progresses this flat face becomes concave, and other concave faces appear. It seems more probable that the first flat face may form at the softest part of the ball surface, and that each succeeding contact tends to force this flat face into sliding contact with a larger round ball; than that the flat faced ball tends to pair off with a particular round ball and to travel with it continuously. When the small worn ball has a flat face and is in sliding contact with a large round ball, the surrounding large balls will assume a more or less definite pattern, and slide against the worn ball, thus producing secondary concave faces. The primary concave face seems to be larger and better developed than the secondary faces.

The ball charge can be divided into "concaves" which show at least one concave surface, "intermediates" which have developed flats or incipient concaves, and "rounds."

Ball slippage is always present in a tumbling mill, and the mutual ball movement is necessarily a combination of rolling and sliding. The sliding motion is apparently concentrated upon the smaller worn balls which nest between the surrounding larger round balls. When each worn ball starts its upward path in the mill its primary flat or concave surface fits against a larger round ball, and the round ball slides upon it. The action may be something like that of the ball separator in a ball bearing, except that the worn sliding balls are always under considerable pressure.

The material is ground under the combined influence of breakage 1—by impacts between falling balls and between falling and supported balls, 2—by being nipped between rolling balls, and 3—by being rubbed between the sliding balls. The rubbing action will be increased in the presence of worn balls with concave surfaces. The rubbing action probably produces a considerable portion of the finely ground slimes in the product.

The worn balls commonly approach tetrahedrons in shape, and are very different from concavex, each of which has two equal opposed concave surfaces. Concavex were designed only to grind upon themselves, and not for use in combination with grinding balls. Their action in a grinding charge is very different from that of the concaves, or small worn balls with concave surfaces.

Laboratory grinding tests should give an indication of the change in efficiency caused by the presence of the concaves. A series of such laboratory tests has been planned.

There can be no question but that grinding efficiency should be considered in the light of overall plant results rather than from the standpoint of the size reduction accomplished. However, no convenient yardstick other than size analyses is presently available by which the effect of a proposed grinding installation upon the plant recovery and reagent cost can be predicted. Until such a yardstick is developed, it will be necessary to consider grinding primarily on the basis of feed and product sizes.

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[3] J. F. Myers and F. M. Lewis: Progress Report on Grinding at Tennessee Copper Co. *Trans.* AIME 187, (1950) p. 707; *Mining Engineering* (June 1950). TP 2863B.

[4] L. E. Djingheuzian: A Study of Present-Day Grinding. *Trans.* C.I.M., Vol. 52: 243-257 (1949).

Flotation Rates and Flotation Efficiency

by Nathaniel Arbiter

DISCUSSION

T. M. Morris *(School of Mines and Metallurgy, Rolla, Mo.)*—Rate studies promise to help quantify flotation operations. The author's exposition of rate studies is therefore laudable. However his exposition of a second order rate equation rather than a first order rate equation is not convincing. It is important to use the correct equation, otherwise erroneous conclusions may be drawn.

The data assembled by the author and plotted according to a second order rate equation can be plotted according to a first order rate equation. As a matter of fact much of the data used reveals that a second order rate equation is not applicable. According to the method used by the author for plotting data, the reciprocal of the slope of the line is the maximum possible recovery. Physically this cannot exceed 100 pct, yet several graphs yield maximum recoveries greater than 100 pct. For example: 1—Curve 2, Fig. 1 yields a value of 120 pct; 2—Curve 2, Fig. 2 yields a value of 115 pct; 3—Fig. 7 yields a value of 110 pct.

The real test of the graphical method is, as the author points out, whether or not the earliest points agree with a straight line. The second order rate plot does not fulfill this requirement. Fig. 10 shows the data used by the author to obtain Curve 2 in Fig. 1, plotted according to both a second and first order rate equation. Fig. 11 shows the two rate plots made from data published by Ludt and DeWitt.[19]

In Fig. 10 the first order rate plot gives a very good fit and obeys the boundary condition of passing through the origin. For the second order rate plot a straight line can be drawn through the last three points but the *earliest* points are not in agreement with this line. The reciprocal of the slope of the straight line gives a value of maximum possible recovery of 112 pct. The maximum possible recovery according to the first order rate plot is 78 pct.

In Fig. 11 the first order rate plot again fits all of the points very well and passes through the origin. The second order rate plot shows wide deviation of the *earliest* point, indicating that the second order rate equation is not applicable.

The reason for the *fit* of data to the second order rate equation during longer flotation times is that R, the cumulative percent recovery, changes slowly toward the end of the flotation operation, so that when t/R is plotted against time, one is, in effect plotting t against t and one would certainly expect a straight line.

The author attempts to show that the second order rate equation is correct by citing evidence which shows that recovery is a function of initial concentration. This is a debatable point especially since, in the author's original presentation of this paper at the St. Louis

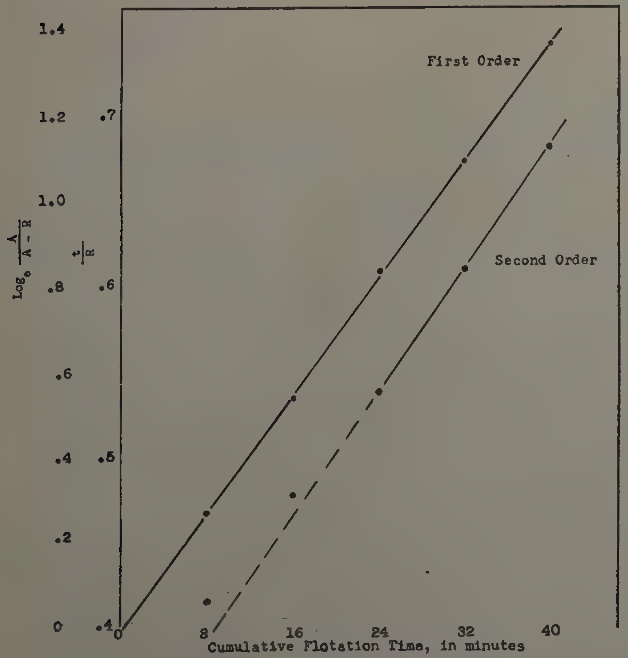

Fig. 10—Data of Zuniga, gold tailings.

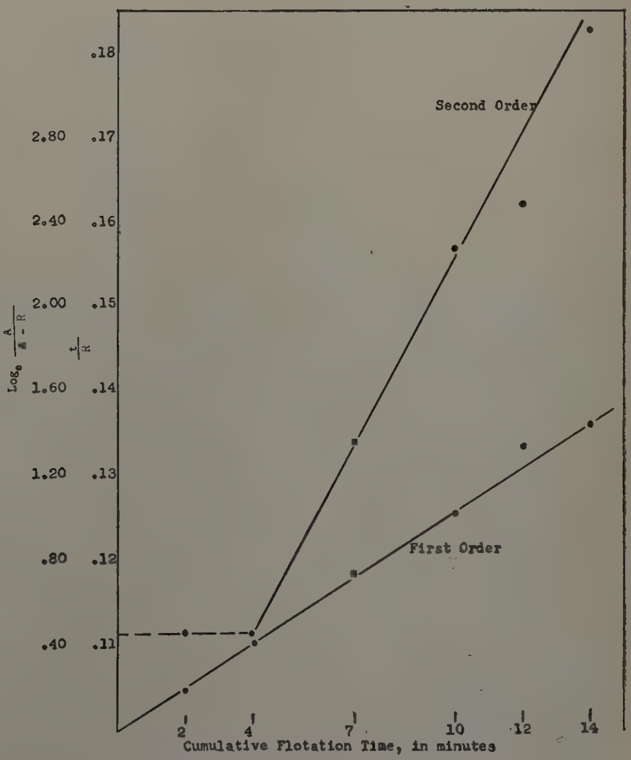

Fig. 11—Data of Ludt and DeWitt.

meeting of the AIME, he included a table showing that recovery was essentially the same when the pulp density varied (which is similar to variation in initial concentration).

We have been conducting rate studies at the Missouri School of Mines and Metallurgy and find that a first order rate equation is applicable to our data.

Nathaniel Arbiter (author's reply)—The criticisms raised by Mr. Morris consist of the following: 1—that for one out of 18 sets of data in the paper, and for one additional set, a first order equation gives a better fit than a second order; 2—that for three of the sets of data, one of the constants in the equation does not have real physical meaning.

Granting the correctness of the criticisms, I do not see that they affect the thesis of the paper. The paper does not advance the second order rate equation as exclusively applicable. Rather, it seeks to demonstrate the utility of the rate approach in understanding the flotation process.

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[19] R. W. Ludt and C. C. DeWitt: The Flotation of Copper Silicate from Silica. *Trans.* AIME (1949) 184, p. 49; *Mining Engineering* (February 1949). TP 2535B, Table I, Experiment No. 36a, b, c, d, e, and f.

A New Method for Recovery of Flake Mica

by R. Adair, W. T. McDaniel, and W. R. Hudspeth

DISCUSSION

O. C. Ralston *(U. S. Bureau of Mines, Washington, D. C.)*—Flake mica can be beneficiated by a wider variety of methods than almost any known mineral. However, most of these methods are not recorded. It is therefore a pleasure to find a written record of another application of a new method of beneficiation.

The fact that the Humphries spiral gives a concentrate containing the mica flake mixed with the slime and small grit has forced a further separation by screening, in which the finest of the flake will pass through. Fine flake has its uses and if mixed with slimes or clay is recoverable reliably only by flotation. This loss must be charged against the spiral. On the other hand, the new muscovite mica paper recently brought out by General Electric Co., can be made only with grit free flake larger than 50 mesh. It is of interest to learn from the authors whether the washing of the flake on a 50 mesh screen can take the grit content below 1 pct, to meet this new demand. Must another degritting process be used on the flake, like electrostatic treatment of the dried material?

H. D. Snedden *(The Humphreys Investment Co., Denver)*—When recovering mica in the spiral, gangue particles coarser than 16 mesh can be rejected more readily when operating at a high pulp density (35 pct solids) or a low volume rate (—15 gpm).

A launder screen as used by Lehigh Coal Co. has proved very satisfactory to partially dewater the mica concentrate and to reject any gangue slimes it may contain.

Under certain conditions mica cleaning in the spiral can be improved by reagentizing the spiral feed, the object being to insure wetting of the gangue minerals and to agglomerate or prevent wetting of the mica particles.

R. B. Adair (authors' reply)—The grit content of the mica recovered on Humphreys Spirals varies with the fineness of the launder screen used to clean the mica product. Normal practice has been the use of 80 mesh screen, which will produce a mica product of acceptable grit content for most purposes. By using a 50 mesh screen, and increasing the screen area per ton of feed, it is felt by the authors that a mica of 99 pct plus purity could be obtained. This statement would hold only in the case of mica feed which was well delaminated, and free from "clay balls," (semikaclinized feldspar), which tends to follow the mica through the process.

Flotation Tests on Korean Scheelite Ore

by Will Mitchell, Jr., C. L. Sollenberger, and T. G. Kirkland

DISCUSSION

R. S. Handy *(Santa Rosa, Calif.)*—It would be interesting to learn the comparative results of treating the Korean scheelite ore described in this paper according to the following procedure: 1—Follow the procedure described in the paper to the recovery of the sulphides by flotation. 2—Dilute the residue from the sulphide flotation strongly with water and disperse the contained colloids by adding a mixture of 75 pct sodium silicate and 25 pct quebracho to dispersion point. 3—Settle the dispersed pulp and decant the supernatant colloids. 4—Dilute, settle, and decant the pulp until the settled residue is crystalline. Not over 10 pct of the tungsten minerals should follow the colloids. 5—Treat the settled residue by flotation, having established the proper pH. Oleic acid homogenized with cresylic acid or kerosene is more selective for scheelite than is oleic acid alone. (The recovery of the contained tungsten minerals should be practically complete in this operation.) 6—Deflocculate the tungsten concentrate from step 5 with the same dispersion agents as in step 2 and treat the deflocculated pulp by gravity. The minerals that tend to float with scheelite are generally of low specific gravity and the separation is efficient. The heavier contaminants, such as pyrrhotite and magnetite can be removed magnetically, leaving a practically pure scheelite.

E. J. Pryor *(Imperial College of Science and Technology, London)*—It occurred to me that instead of using temperature as a differentiating control, more precision might be obtained with a reagent of a type we developed here for the flotation of high grade fluorspar in cold pulps.

This reagent, which is manufactured on a modest scale in England, is based on oleic acid, which has been modified by our patented process to ensure complete stable dispersion at any desired concentration in water. It thus becomes possible to utilise unimolecular layers with good selectivity.

In the fluorspar flotation for which we developed it, it is possible to hold a tight control on depressing quebracho for the calcite without substantial loss of fluorite, and recovery exceeding 99 pct is not uncommon in the mill where it is in use, even under wintry conditions.

No attempt has been made yet to extend the use of the reagent, as we have not completely overcome manufacturing and packaging difficulties, but it has shown good results on numerous fluorspars from various parts of the world, and we are extending the tests to other nonmetallics now floated by straight oleic acid. It is not easily upset by hard water, though for laboratory tests distilled or deionised water is desirable in the first place.

If you wish, I will send you a small free test quantity. Until we have succeeded in making it as a solid, commercial application seems to us limited. Cost should ultimately vary up to 30 pct above that ruling for oleic acid.

Will Mitchell, Jr. and **C. L. Sollenberger** (authors' reply)—The purpose of this investigation was to determine the effect of several variables encountered in nonmetallic flotation on the recovery and grade of scheelite in rougher concentrates and did not intend to imply that the combinations used were considered ideal for maximum recovery and grade in cleaned concentrate.

We have investigated the procedure suggested by Mr. Handy and have found that about 10 pct of the scheelite was lost in the desliming operation. With a sodium silicate-quebracho mix a recovery of 76 pct of the scheelite was obtained in rougher float at a grade of 18 pct. With additional depressant in the cleaner float, an increase in grade to 22 pct was obtained but much of the coarse scheelite did not float; the recovery in this operation dropped to 36 pct. The final cleaned flotation concentrate from the procedure consisted mainly of fluorite and scheelite. Since the ratio of these minerals in the head sample was about 2 to 1, the same ratio was obtained in the cleaned concentrate. This concentrate would have to be further treated for removal of fluorite by flotation as described in the paper or by tabling as suggested by Mr. Handy.

Mr. Pryor kindly sent us a sample of the reagent mentioned and we tried it on the ore. No spectacular results were obtained although the reagent was a good collector for scheelite. Unfortunately the ore contains about twice as much fluorite as scheelite and the reagent does not aid this separation.

F — Coal

Coal Preparation in England and Holland

by John Griffen

DISCUSSION

K. F. Tromp *(Kerkrade, Holland)*—Your assumption that the Dutch State Mines have lead in the development of heavy medium processes—the Barvoys, Loess, Driessen—is not correct. The credit should be given to a Dutch engineer Mr. deVooys, the inventor of the Barvoys process.

deVooys started in 1932 in Hückelhoven, a German but Dutch-owned mine, where he was a director. Several plants on the Continent as well as in Great Britain were erected.

The State Mines started their first plant in 1937. They could not get that plant working satisfactorily and built another one. Their process made no headway, and up to 1946 there was only one plant installed, the one at their own pit. Since 1946 they started the cyclone process and only since that time have they been more active, though industrial cyclone washeries are still in their infancy.

deVooys sold his patents to the Dutch State Mines in 1946 and this might have given you the impression that the pioneer work had been done by the Dutch State Mines.

John Griffen (author's reply)—I wish to thank Mr. Tromp for giving the correct story on the development of heavy medium processes in Holland.

Factors Influencing the Choice of a Loading Machine

by D. W. Mitchell

DISCUSSION

J. H. Schlobohm *(Joy Manufacturing Co., New York)*—This paper has been read with a good deal of interest; however, there are several salient features which Mr. Mitchell has overlooked.

The initial emphasis on selection should be placed on type of material to be loaded and method of mining involved.

There is no possible connection in the comparison of scraper or Joy Lohite loaders with loaders of any other type. Neither can an operating comparison be made between "Rocker type" loaders and those equipped with loading head and gathering arms. Each unit has its application, but the application depends on material and the mining system involved.

Mr. Mitchell's compilation of data on various loaders, while it represents a great many facts, does not represent true operational data, i.e. Table I, column heading "Max. Rated Capacity for 7 hr;" these figures are meaningless inasmuch as little or no time at all is allowed for tramming or switching or moving the loader from place to place. Table I, column heading, "Best Reported Field Shift," would be a better source of operational data, assuming of course that conditions, applicable to each class of loaders, were average to avoid enthusiastic performance figures.

Table I, column heading, "Repair Parts Cost per Ton," is still another item which will not stand up under comparison unless the loaders involved are put into type classes, i.e., there are simply no means possible to show the repair costs of scraper loading versus a caterpillar mounted loader in abrasive material. The source of information here is also important. Actual field tests have proved that these costs are of a constant nature. Manufacturers are striving to hold them to a minimum and many interesting studies have been made. A fairly recent example of this in a large United States gold producer has proved that Joy HL-3 loaders can be consistently average, less than $0.025 per ton mucked. This cost represents repair labor and parts over a two year period in heavy ore. Again, however, no real comparisons of repair costs can be made without taking the actual working conditions into consideration. It has been found repeatedly that many operators are perfectly satisfied with high maintenance costs to achieve required tonnage. These high costs may be caused by poor bottom, power conditions, or even minor maintenance neglect but all of these factors are justifiable if the end is accomplished. This is particularly prevalent on high speed contract work.

Many of the loaders listed are typical "hard rock" loaders, particularly the Joy Model 18HR2 and the Goodman Conway loaders. The Joy 18HR2 has a rated capacity of 6 to 12 tons per min and is by far the fastest loader available on the market. This machine is now being successfully used in many mines in the United States as well as abroad and while we know that tonnages of over 900 tons per shift have been mucked, average tonnages run about 500 to 550 tons per shift. These figures are based on tonnages of iron ore and would not be applicable in high speed tunnel work where the Joy 18HR2 is perhaps best adapted. Here, in large bore tunnels where deep rounds can be pulled, this machine is chalking up remarkable tonnage records.

Based on the aforegoing it is felt that a more realistic view of the problems be taken into consideration before any actual choice of loading machine be made. The operator should under any circumstances arrange to see loaders working in conditions which are similar to his own. Then, after careful study, he should know which machine is best adapted to his particular circumstances. Consideration should be given to the factors influencing the choice of a loading machine in the following order:

1—Material to be loaded, 2—Mining system, 3—Required tonnage, 4—Maintenance cost, 1 + 2 + 3 + 4 = *class of loader* to be used.

It is granted that there are many more factors and even subfactors, but these basically should be given primary consideration. Regardless, however, of what selection is made, a mine loader is only as efficient and productive as operating conditions permit. Transportation of material as well as location of working places are too often neglected factors in loading problems and without proper auxiliary equipment full productive power cannot be reached.

In the equation above the term *class of loader* is italicized. The operator is now ready to take into consideration such factors as maximum dimensions, power means, type mounting, etc. The importance of these features is minimized because it is felt that a selection can be made from the wide range of available machines, and to sacrifice the advantages of a minor factor will in all probability lose its importance when high tonnages are attained. The manufacturers are more than willing to consider special applications and in most cases standard loaders of one class or another can be utilized. In any event, the operator will do well to consult with manufacturers or their representatives concerning problems which involve machinery.

Efficiency and Sharpness of Separation in Evaluating Coal-Washery Performance

by H. F. Yancey and M. R. Geer

DISCUSSION

John Griffen *(Pittsburgh)*—I wish to congratulate the authors on this paper, which, I am sure, will promote a clearer conception of the various criteria which have been advanced as measures of coal-cleaning performance. Their clear descriptions of the more important criteria that have been advanced and explanations of their meaning and application have been sorely needed and, if widely promulgated and followed, should remove much of the confusion which, heretofore, has been evident.

I suggest that much of this confusion has also arisen from lack of common agreement as to the exact meaning of the words and terms used in our discussions of this subject. I am sure I have been equally guilty in this respect and suggest it is time that we select a committee to study vocabulary as well as criteria, so that common understanding and agreement can be had.

Thus, I suggest, that the title of this paper should have used the words "coal-cleaning performance" and not "coal-washery performance." My understanding is that "washery" refers to a complete plant and it has additional functions to that of coal-cleaning, such as control of size, moisture, etc.

Also, I do not agree with the authors that the formulas given in their paper, the Fraser and Yancey formulas, are necessarily the only ones that can be called efficiency formulas. When a dictionary is consulted one realizes that the words "efficiency" and "efficient" have not precise meanings but only very general ones and must be further defined for each particular use, if they are to have any precise meaning.

To my mind, the first of the Fraser and Yancey formulas given should never be used as expressing "efficiency" alone but only by the fuller phrase which truly expresses the kind of efficiency it states, which is "coal recovery efficiency." For complete clarity we then need only define "coal" for the particular circumstances under which the formula is used. This would recognize the fact, fully explained by the authors, that the limits in coal qualities imposed by the market have imposed a factor—and a variable factor—which has influenced any answer obtained by this formula. Such a procedure makes it entirely practical and logical to use the values of any of the qualities of coal—ash, sulphur, phosphorus, specific gravity—or a combination of them to define what is coal for that particular use of the formula.

Further, it is clear to me that the criteria to be used must be dictated by the objective in view and the use for which the data are obtained. The authors clearly point out that the Fraser and Yancey coal-recovery efficiency formula is influenced by three factors, "one is determined by the coal, one is dictated by market or use conditions, and one is an inherent characteristic of the cleaning unit itself." The formula which uses the inverse of misplaced material as a measure of efficiency is also a coal recovery efficiency formula and is influenced in like manner by these three factors. As the authors point out, both are useless in obtaining data which enables a direct and valid comparison of the performance of a given cleaning unit on two coals where the other two factors vary or of the performance of two or more cleaning units of different types unless the values of the other two factors are identical. Much of the confusion in our discussions of and literature on coal cleaning has come from failure to understand clearly this fact. Many such comparisons have been made from which erroneous conclusions have been drawn.

I am, therefore, particularly glad to see the authors' emphasis on the criteria which indicate the "sharpness of separation." I feel that these are the essential criteria in which the coal cleaning fraternity should be primarily interested—technically and practically. The characteristics of the raw coal were determined by nature and the mining organization, while the qualities of the washed coal are dictated by market requirements and the opinions of the coal sales department. These two factors are seldom subject to change. Thus, the sharpness of separation of the various types of coal cleaning equipment is the only one of the three factors over which any control can be had. It is only by the proper selection of cleaning units having the necessary sharpness of separation to produce the product the market requires from the raw coal available, that the optimum coal-recovery efficiency can be obtained. The use of the coal-recovery efficiency data as the only guide in selection, when the other factors are dissimilar, confuses the issue and has resulted in unhappy selections.

Even the use of the various criteria to determine sharpness of separation seems to need further exploring before they can be used as precise tools. We need far more data on these criteria for each type of coal cleaning unit, which shows the effect of size range and of variations in throughput, as well as the gravity of separation, on the sharpness of separation. In like manner we need data which will indicate the characteristics of each type of cleaning unit as to the cleaning performance on individual sizes of coal of a wide size range being cleaned in one unit. This is of vital importance when such individual sizes are to be marketed as such.

It has been shown by the authors and others that the various coal-recovery efficiency formulas are ineffective for the purposes described in the preceding paragraph and only the criteria which evaluate density and sharpness of separation are useful. Therefore, it seems to me the authors beg the question when they stress the greater amount of test work, including a wider-range of testing densities reaching to higher gravities, required to evaluate properly these criteria. It is only by conducting our investigations thoroughly and accurately that we can arrive at valid conclusions. Therefore, I feel confident that the test work to be done for these purposes must be safeguarded by adherence to all the details which the Europeans have found essential. It is my understanding that the most important of these conditions are:

1—Distribution curves and any criteria derived from them must be based on a reconstituted feed, to eliminate any errors arising from degradation or crushing of the feed material during treatment and from sampling inaccuracies.

2—A sufficiently wide range of testing gravities must be used so that the distribution of the coal of lowest gravity and the refuse of highest gravity can be determined within a probability of error that would be insignificant. I believe K. F. Tromp claims that the values obtained for these criteria are questionable, unless the distribution curve can be closed at both ends.

I offer as an additional suggestion that the reporting of distribution data on a fine size reaching to zero is very questionable, certainly for equipment utilizing gravity as a means of separation. We feel quite sure that at some small particle size differences in specific gravity become an insignificant factor in effecting separation. Further, on such fine materials, the separations made by the testing liquids are far from accurate unless centrifuges and elaborate retreatment are employed.

Referring to the subject matter of condition 1 above, I would say that failure to observe this requirement has been most glaring in this country and has resulted in the presentation of performance data which is very definitely misleading. The paper presented by R. E. Zimmerman[17] contains an illustration. In his Fig. 8 he plots the cumulative float ash data of the feed sample and of the cumulative table zone products. A comparison of these curves shows a very high coal-recovery efficiency for separations at 1.50 sp gr and higher densities. However, the feed sample data shows a total of 20.8 pct ash while the total of the table zone products is only 19.5 pct ash. The products were tested at only 1.5 and 1.6 sp gr and a calculation of the reconstituted feed at these two gravities shows a wide divergence from the amounts and ash contents of the feed at these points. When the cumulative zone products are compared with the reconstituted feed, the amount of misplaced material is doubled and the coal-recovery efficiencies in this ash range of clean coal are very significantly reduced.

As the authors point out, the error area may not necessarily have a fixed relation to coal-recovery efficiency. I suggest, that if a system be devised that will properly weight the respective error areas according to their distance from the density of separation, a term could be obtained that would bear a much closer direct relation to coal recovery efficiency. Whether it would provide a more useful criterion is uncertain.

The definition of the specific gravity of separation as that at which the distribution curve crosses the 50 pct line if considered alone may result in wrong assumptions. On page 514, the authors point out that the wet table distribution curves for different size ranges show a trend opposite to that of jigs, in that, "with the wet table, decreasing particle size is accompanied by decreasing density of separation." Reference is made to Fig. 8 which confirms the statement as made. Normally, one assumes that a separation at a lower density should result in a washed coal of better quality. However, if one examines these distribution curves, one notes that a feed size range of 8 mesh to 0 has been examined in three sizes, 8 to 20 mesh, 20 to 48 mesh and 48 mesh to 0. The density of separation, as defined, progressively decreases but the error area of the 48 mesh to 0 has increased so much that it is very likely that this size of the washed coal is higher in ash than either of the other sizes. It is probable that the 20 to 48 mesh washed coal, which shows an intermediate density of separation, has the lowest ash content and shows as sharp a separation as any of the three sizes.

In conclusion, I feel our situation is similar to that of the statistical presentation of data, where no single term, such as average, average deviation, etc., fully defines and represents the facts; but a combination of a number of terms is required. To provide tools which we can use to evaluate coal-cleaning performance properly, we also need several terms or yardsticks, so that we may advance the art of designing and operating such plants to obtain optimum results with minimum costs.

I feel the authors' paper will prove to be a most important guide to a proper appreciation of the conceptions so far advanced and in educating the coal industry to their proper use and they are to be highly complimented. The crying need is for complete and accurate test data in great wealth which can be evaluated by these various yardsticks so the capabilities of coal cleaning equipment can be known under all conditions. Coal has become too valuable to be wasted by the improper application of equipment which under different conditions has done an efficient job.

W. W. Anderson *(Commercial Testing & Engineering Co., Chicago)*—Messrs. Yancey and Geer have prepared an exceptionally thorough and scholarly review of the better measures for evaluating coal washery performance. In addition to presenting discussion of the usefulness of various measures of performance, the paper contains several important conclusions not found elsewhere in American literature, particularly in regard to the worth of error areas and probable errors.

Apparently a number of investigators have been motivated by the casual resemblance of the curves of distribution to the Ogive, or cumulative frequency curve of statistics, to hope that distribution curves would behave in accordance with the laws of probability. Numerous writers have stated that the shape of the distribution curve, with the top half inverted, is similar, except for skewness, to the Gaussian error distribution curve, neglecting to mention that most distribution curves have positive kurtosis, or a peak at the gravity of separation greater than that of a normal curve of probability. Furthermore, the curves of distribution which neither look like the normal curve, nor any other smooth curve of frequency distribution, are usually disregarded. Therefore, it is not altogether surprising that a probability plot of distribution data fails to develop a straight line.

Nevertheless, the idea that the laws of chance may apply at least to some curves of distribution should stimulate efforts, not only to ascertain causes for deviations of distribution curves from probability frequencies, but also to characterize distribution curves by a few arithmetical constants or suitable formulae.

The lack of correlation between error area and misplaced material should not have been unexpected, since error area is obtained from the distribution curve which is a plot of ratios, whereas misplaced material is expressed as a direct percentage of all products.

It is clear from this paper that error area reflects only distribution of misplaced material, rather than the amount of misplaced material. A three-dimensional value, such as error volume, is needed to express quantity and distribution of misplaced material. After all, the surface area of a lake does not indicate the volume of water in the lake unless the additional factor of depth is known. Why any of us should have expected error area, as currently determined, to measure efficiency, or the amount of improperly distributed material, is more astonishing than proof to the contrary.

More than a year ago, W. L. McMorris of the H. C. Frick Coke Co. suggested to the writer that the abscissa scale should not be laid out in equal intervals of specific gravity, but in proportionate intervals of gravity, in accordance with weight distribution of the gravity consist of the combined products. This would appear to be a reasonable suggestion, not only to enhance the value of error areas, but also to present more perfectly the curves of distribution. The writer regrets that he did not have time to prepare evidence for this discussion of the worth of this suggestion; but the idea may have sufficient merit that other investigators would want to study it for themselves.

Plot of the data in Table IV of percent misplaced material versus efficiency indicates a pretty fair correlation, and a similar plot of the data in Table III demonstrates an even better correlation of these two measures of efficiency. Incidentally, it should be noted that Yancey speaks of his measure as "efficiency of recovery," whereas Anderson terms his measure "efficiency of separation," which in each case is proper terminology. Since values determined by either method appear to be of the same order, use of either method should not lead to confusing interpretations, except in the case where no true separation has occurred. In such a situation, Yancey would report 100 pct efficiency of recovery, and Anderson 0 pct efficiency of separation.

Undoubtedly there are occasions for the use of one method in preference to the other for rapid and inexpensive determination of washery efficiency. For instance, if the float and sink data on the feed are available, the recovery method is the quickest approach to efficiency. However, when feed data are not available, the most rapid method to ascertain efficiency is from the amount of misplaced material. This latter method is also the cheapest when appreciable size degradation occurs in the washing process, because then Yancey's method cannot be used safely without complete analytical data on the washery products for calculation of a composite product gravity consist.

Whenever Yancey's efficiency of recovery is applicable, the user should remember that Yancey's method tends toward high values of efficiency because the method credits size degradation to washery efficiency.

At some cleaning plants preparing metallurgical coal, where size degradation means no loss in sales realization, the high values of efficiency determined by the Yancey formula are probably of no great consequence. In contrast, many operators of cleaning plants specializing in preparation of stoker coals are vitally interested in not being misled concerning washery efficiency and size degradation, because these items directly influence sales realization.

A minor point to which exception is taken is the assumption as stated in the following quotation from the paper: "European practice apparently involves the use of more and higher densities in examining samples for performance tests. Thus, American practice may have to follow the European system . . ."

Clients of the Commercial Testing & Engineering Co. know that performance tests of their cleaning plants include a sufficient number of gravity fractions to properly locate the curves of distribution; and if fine sizes are under study, densities up to 2.17 are incorporated in the test procedure. Hence, any assumption that all American practice lags behind European methods is false.

In conclusion, the writer feels that American coal preparation engineers should be grateful to Messrs. Yancey and Geer for their excellent paper, which not only reviews the Paris meeting of the First International Conference on Coal Preparation; but in addition, and what is more important, it clarifies some of the measures of performance which have been in sad need of critical study and unbiased evaluation.

G. A. Vissac *(Seattle, Wash.)*—Coal preparation in the United States has shown remarkable progress and advances during the last few years. As a consequence there is now a real demand for more scientific methods of controls, and the paper of H. F. Yancey and M. R. Geer will be appreciated by all washery engineers.

It would be desirable to unify terms and methods. My own solution has been to adopt terms and methods now in general use in this country, in connection with other industries, sciences and activities.

There is nothing new in the mathematical analysis now applied to coal washing; use is made of the well-known laws and theories of chance, or great numbers, or probabilities, fully described in all technical books dealing with statistics.

Many terms such as dispersion, mode, frequencies, probable error are defined; I submit they might be adopted by the coal industry, just as they have been adopted by many other branches.

I suggest we should also adopt the same methods and standards of graphic presentation; namely, the straight line presentation commonly used for similar problems, and for which standard graph papers are available in this country (Codex Co., Norwood, Mass.).

The advantages of logarithmic scales to present such distributions are obvious. However, we must learn how to use, or adapt, these standards, methods and graphs.

First, tendencies only must be looked for; not accurate lines, but the nearest lines covering the events most likely to happen.

The mathematical laws we are using are actually called laws of great numbers, in other words, they will only best cover the most common results out of a large number of tests.

Tests must be kept within practical range; for instance, densities at 1.3 or 2.2 are of no practical value in most coal cleaning problems.

The value and purpose of the distribution curve, or line, is to evaluate as correctly as possible, but relatively, three main characteristics; namely, the actual gravity of separation and the two probable dispersions.

As the first characteristic is found at the 50 pct distribution, and as the other two are found respectively between 25 pct and 50 pct, and 50 pct and 75 pct respectively, it is obvious that the only interesting part of the distribution curve is between 25 and 75; and the points beyond are of very little value, and the main thing is to concentrate the tests on the results between the above critical points.

Origins and scales must be carefully selected. The most common errors in this connection are due to the fact that sufficient distinction is not always made between what is measured and what is analyzed.

The ash content of a product is made of a constant: the inherent ash characteristic of this particular coal and size, plus a variable; the impurities, or removable ash; and this is the only variable to be considered if we are to use logarithmic representations correctly. This can be translated in gravities, namely, gravity of the pure coal, gravities of the foreign materials.

However, gravities must be considered as a "mean" only, but the final practical results must always be expressed in ash contents.

Pure coal, inherent ash must be understood at their practical value, namely, for each particular size of each particular coal, it will be the float at the low practical density; namely, from 1.30 to 1.40, or to be more precise, the density of separation less — 0.20, or — 0.15.

It is well known that the various sizes of the same coal have different sink and float characteristics. Furthermore, most of the new methods of control have recognized the influence of size on results obtained with a given machine and a given coal; still many tests used for demonstration purposes do not take sizes into consideration, and accordingly are of no practical value.

In all cases, to be able to compare results and to interpolate tendencies, a definite and narrow size ratio must be dealt with, and the ratio of 2 to 1 suggested as narrow enough to give consistent and scientific results; namely, the following sizes: 2 in.—1 in., 1 in.—½ in., and ½ in.—1.4 in.

Results of operation on such wide ranges as say ¼ in.—0 are not suitable to scientific analysis; and trying to compare an operation on say ¼ in.—0 size to an operation on say ¼ in.—1 in. is absolutely without value.

On the above scientific basis only can we hope to approach accurate and useful methods of control. Existing operations must be analyzed first to determine and analyze the parameters and characteristics involved.

They can be classified in two groups; 1—Factors proper to the washing machine when dealing with various sizes, various coals, and various densities, and 2—Factors proper to each particular size of each particular coal.

Obviously, the results of a great many tests must be analyzed and put in order in simple and workable forms, then translated the results in simple "empirical laws." As a result we will then be equipped to tackle the next problem, namely, the scientific design of a new preparation plant.

Only by the application of such scientific methods can we expect to clarify the situation as it exists today. New methods, new machines, are brought up (or unearthed); new and more stringent requirements are imposed; lower grades of coals have to be considered; it becomes more difficult to make a correct selection between the many solutions available today.

However, as a matter of fact, there is no such thing as a better machine, or a better process, but there may be a more desirable machine or process to suit each individual case.

On the basis of market requirements and characteristics of the raw product, a scientific approach only of the problems involved will supply the best and most economical solutions. We must be grateful to Cerchar and Yancey and Geer for their outstanding work in this connection.

Arthur Grounds and **L. W. Needham** *(National Coal Board, London)*—The authors have been careful to draw a distinction both in the title and in the paper itself between the sharpness of separation and the efficiency of the cleaning process in particular cases. It will be possible to make more rapid progress with regard to the proper evaluation of processes when this distinction is much more widely understood than it is at present. Attempts to express the sharpness of separation are, broadly speaking, attempts to express the characteristics of cleaning processes. The efficiency of separation, on the other hand, is an expression of the effect of the characteristics of processes on the products of separation when particular coals are treated.

The suggestion underlying many contributions to the subject, notably those by K. F. Tromp, is that the sharpness of separation characteristic of certain processes is, to a very large extent, independent of the composition of the raw coal feed. This suggestion is often received with considerable scepticism simply because it is not properly understood. Yancey and Geer, however, make very useful comments about it, especially on page 514 in the paragraph headed "Influence of Density Composition of Raw Coal." Clearly, if there are characteristics of processes which are independent of the density composition of the raw feed, and these can be understood, the results of separating any particular coal of known composition can be calculated beforehand with some accuracy.

Characteristics of processes may not be important if different methods of treating the same coal are compared because it may be sufficient simply to compare the results obtained. On the other hand, if it is desired to know how a given process would deal with a variety of coals, its inherent characteristics are of fundamental importance.

Yancey and Geer's paper is a valuable contribution to clear thinking on these points.

It is true that there is at the moment a lack of experimental evidence to decide how far characteristics of processes, independent of coal composition, can be justified. The experiments referred to in the paper should give very valuable information. In the meantime, it may be suggested, on the basis of experience with British plants, that the distribution curve is probably substantially independent of the composition of the raw coal for dense medium washers, especially those using a comparatively stable medium in which currents are at a minimum, but that in Baum or trough washers, where the bed or "separating medium" is made up of selected parts of the feed, it is much less likely that the distribution curve is independent of the density composition of the raw feed.

Fig. 4 is of particular interest and may be compared with the graph which we have prepared, Fig. 11, and which shows the relationship between error area and "écart probable" for a number of separations in British plants. The point shows a good distribution about a straight line as distinct from the curve as suggested by Yancey and Geer's results. Incidentally, we should like to endorse the comment made that the "écart probable" does not reflect directly the disposition made by washing of the lightest and heaviest fractions of the raw coal. From this point of view the error area has advantages as a means of expressing distribution.

Results obtained on British plants confirm the tendency noted for separation gravities, error areas and "écart probable" values to increase as the size of the particles separated decreases. The exception quoted, for separations on a concentrating table, is interesting and it would be instructive if the authors could discuss further the possible explanations which must depend on some conditions affecting washing on tables quite differently from other processes.

The section dealing with the influence of size composition is concerned simply with noting the observations on different sizes of products, but does not discuss the effect of size composition as such on the overall separation resulting from washing operations. How, for example, would the separation produced in 3 in. — 0 feed of normal composition differ from that obtained if the feed were deficient in the fine sizes? If the authors have any experience to enable them to comment on this point, it would be helpful.

There is only an incidental reference to the ash error method of expressing the results of a cleaning operation. In Great Britain, this method is often used because it expresses the effect of the misplaced material on the ash content of the clean products which is, after all, a property of great importance to the customer who buys the coal. Possibly the ash error does not add

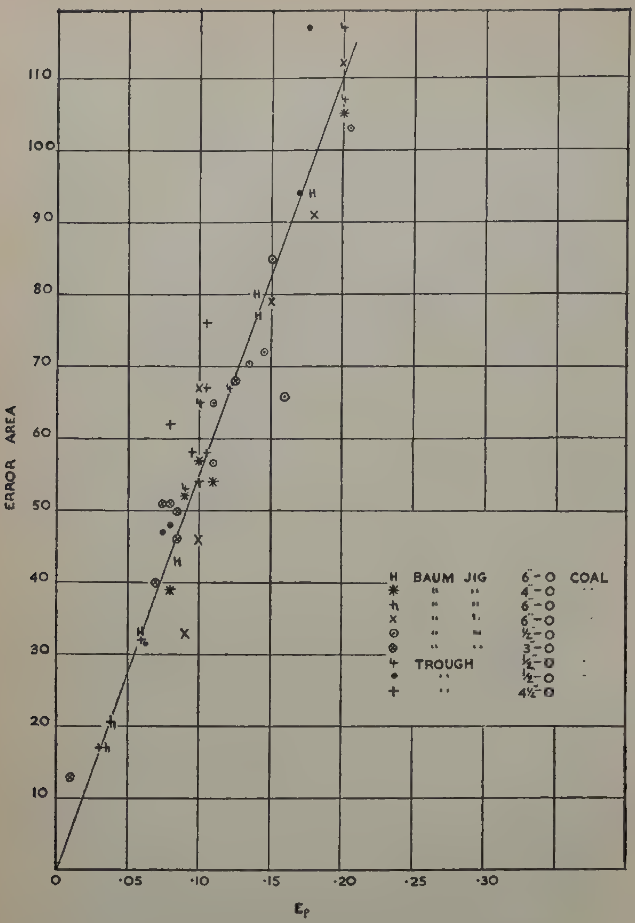

Fig. 11—Relation between error and "ecart probable."

very much to the value of distribution curves and other data illustrating the accuracy of separation from the point of view of a coal preparation engineer well versed in the various processes and methods of expressing results, but from the practical point of view it seems obvious enough that a process which can give a clean coal with an ash content differing very little from that theoretically obtainable for the same yield is more accurate than one which shows a much larger difference between the actual and theoretical ash contents. Moreover, the relation between ash error and yield error[18] enables the significance of the more accurate processes to be understood easily, especially when low gravities of separation are required. It would be interesting if the authors could comment on the use of the ash error as a means of summarising the total effect of the errors of separation.

Now that dense medium plants are being installed in relatively large numbers in the United States, it is to be hoped that much more information will be collected and published as to their performance characteristics.

K. F. Tromp *(Kerkrade, Holland)*—A number of papers on the subject of evaluating washery performance have been presented recently, but I believe the present report is the best that I have read. The authors have treated the subject in a sound and understandable way, constructively backing up their ideas with experimental results. Some of their ideas, however, are not in agreement with my own.

My concept of the distribution curve, described originally in Glückauf in 1937, is that it is composed of two parts divided by the specific gravity of separation. One part represents coal misplaced in the refuse product, while the other represents impurity misplaced in the clean coal product. The two parts are limbs of two error frequency curves which have not necessarily the same error areas. This requires some explanation. It is common knowledge that the sharpness of separation decreases when the velocity of flow through the washing unit is increased. The shorter the time available for the actual separation, the poorer the sharpness of separation, the greater the error area. This not only applies to the distribution curve as a whole but also to the constituents individually: when there is a difference in the length of time that the coal and refuse products remain in the washing unit, and mostly there is, there will be a difference in error area between the two limbs of the distribution curve.

The authors' tests on the Driessen cyclone provide an excellent example of this relation between the error areas of the two limbs of the distribution curve and the time available for the separation. Doubtless three different orifices for the spigot of the cyclone were employed in the three tests, and the total amount of medium fed to the cyclone probably remained about constant. Under these conditions, the sink or refuse material will have remained in the cyclone a bit longer in the tests made at 1.75 sp gr than in the one made at 1.56. Conversely, the float coal in the tests made at 1.75 will have remained a shorter period of time than in the test made at 1.56 sp gr. According to my theory, the error area of the misplaced float material should decrease when the float material remains longer in the washing unit, and the same applies to the misplaced sink or impurity. This is exactly the relationship shown in the following tabulation taken from Fig. 10:

Specific Gravity of Separation	Error Area Misplaced Floats	Error Area Misplaced Sinks	Total
1.56	18.7	19.3	38.0
1.63	18.1	22.9	41.0
1.75	13.7	29.4	43.1

The authors contest my thesis that the sharpness of separation decreases with increase in the density of separation. However, their cyclone tests show that the quotient of total error area and separating density is practically constant, being: 38.0 : 1.56 = 24.4, 41.0 : 1.63 = 25.1, and 43.0 : 1.75 = 24.6. This is exactly the relationship given in my aforementioned paper (Glückauf 1937) for cases where differences in viscosity can be neglected.

The other results presented by the authors in Table III, for the separations at different densities on the wet table, are not comparable because they have been carried out with different separating times. The separations at lower densities were obtained by shortening the path of travel, in other words, by shortening the time available for the particles to find their proper place. Reducing the separating time means increasing the error area. With these tests made on the wet table, two opposing factors exist, decreasing the effective table area gives an increase in error area, but the lower separating density would give a decrease in error area. Moreover, a wet table consists of a multitude of small washing units, and the distribution curve loses its definite form when the washing process consists of a series of rewashing operations.

When introducing the error curves in 1937, I stressed the point that distribution data should be calculated for only a narrow range of sizes. Otherwise, a separation according to particle size is superimposed on one according to density. The importance of this has been underestimated later by other investigators and the authors should be credited for their extensive work and their illustrative examples in demonstrating the great influence of particle size on error area and, in most processes, on separating density.

W. M. Bertholf *(Colorado Fuel & Iron Corp., Pueblo, Colo.)*—The authors have made a thorough study of the "newer" methods of evaluating coal washery performance, and this discussion is of considerable value in assisting one to understand the latest trend of thought on the subject.

It is, perhaps, unnecessary to point out that the use of "independent" criteria of washing performance requires that we work back through the original density consist of the washery feed before we have an undistorted view of the results, except in the highly improbable case of "rectilinear distribution." These criteria appear to be more useful as tools of the trade than as overall descriptive indices.

We are, therefore, inclined to agree with the authors that formulae similar to the Bureau of Mines "efficiency" (e.g. Anderson's complement of misplaced material) are more suitable for evaluating washery performance than the error area or probable error.

On page 509 of the paper a second efficiency formula is shown, involving the ash content at the specific gravity representing the most profitable separation of the coal. It is remarked that this density is difficult to determine, hence this particular formula has not been used extensively.

In the course of an investigation of the most profitable separation of coal used in the production of blast furnace coke, it was discovered that a set of "fuel cost" contour lines (per ton of pig iron) could be superimposed on the yield-ash grid ordinarily used for washability curves. Details are given in the published account[19] and the Fig. 12 is a purely hypothetical example of the use of this system in evaluating washery performance.

Taking into account the cost of the raw coal, washery operating costs and washed coal yield, we can obtain a cost per ton for any possible yield (independent of ash content). From known relationships the quantity of coke required per ton of pig iron for any ash content (independent of yield) can be computed. Combining the two, "fuel costs" for any combination of washed coal ash and yield can be developed. This grid of diagonal lines is superimposed on the rectangular coordinate system of ash vs. yield. As might be expected, the fuel cost surface slopes upward as washed coal ash increases and/or washed coal yield decreases. The yield-ash curve of the coal cuts through the fuel cost grid and reaches a minimum at some particular combination of the two variables. Going away from this point in either direction increases the fuel cost.

Considering the feed curve, which represents results obtainable only with a perfectly sharp separation, we find that the minimum fuel cost is reached at approximately 7.8 pct ash and 87 pct yield. This corresponds to a separating density of about 1.55.

The products curve, for this particular washery, indicates that in practice the minimum fuel cost will be obtained by making a separation at 9 pct ash with a yield of about 88.5 pct by weight. The separating density is 1.7 on the basis of comparable ash contents, 1.6 on the basis of comparable yields and might be 1.8 on the basis of the 50 pct point of the "distribution curve."

Note that with the imperfect separation of the actual operation the cost penalty for over-washing the coal becomes excessive and that one might just as well remove only a small proportion of the heavy impurity as to attempt to wash to 7.5 pct ash.

It is obvious that the difference between the minima on the two curves represents the cost of using an inefficient washery and that comparisons of different washing methods can be made easily for a particular coal on this three-dimensional grid. One then has a direct measure of the economic efficiency of the various methods.

It appears to be certain that captive operations methods such as those outlined above result in a very effective evaluation of coal washery performance. While the same general principles should apply to free operations, it will be much more difficult to assemble and apply the proper cost factors; nevertheless one must not lose sight of the fact that the real reason for washing coal is to reduce process costs. Proper distribution of the rewards for success along this line is a matter for discussion at another time and place.

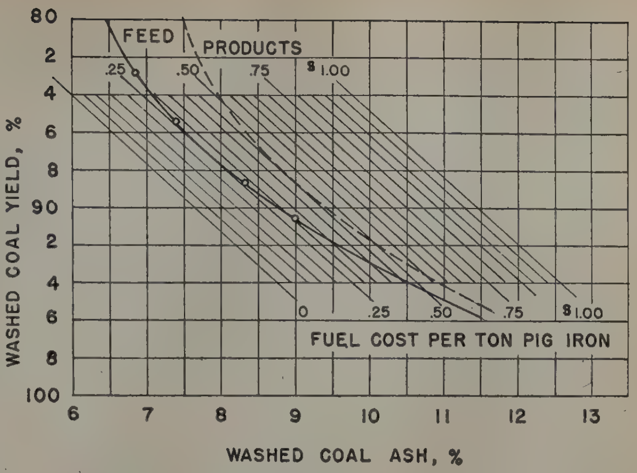

Fig. 12—Influence of washed coal ash and yield on blast furnace fuel cost per ton pig iron.

H. F. Yancey and **M. R. Geer** (authors' reply)—The volume and scope of the discussion of this paper is most gratifying to the authors, for one of their principal objectives was to stimulate the interchange of ideas and information on this important subject.

Both Messrs. Griffen and Anderson stress the importance of determining efficiency from a reconstituted or composite feed rather than a raw coal analysis to eliminate the combined effects of degradation and sampling errors. We have no argument with this thesis, except to point out that under favorable circumstances the results obtained by the two methods of computation are not significantly different. We can scarcely agree with Mr. Anderson's contention, however, that reasons of economy should ever dictate the use of his separation-efficiency formula instead of the Fraser and Yancey formula for recovery efficiency. In the first place, more densities are required to fix accurately the position of a distribution curve than are needed to provide an accurate yield-ash curve. Secondly, making float-and-sink tests of washery products without incurring the relatively small additional expense of determining the ash or sulphur content of the density fractions is stopping short of the goal. The amount of high-density impurity contaminating a washed coal is of limited interest, but the ash or sulphur content of this material is vital to both the sales department and the consumer. Similarly, the low-density fraction of the refuse is of interest to the operator only insofar as it indicates a loss of coal of saleable quality—and saleable coal is reckoned in terms of ash or sulphur content, not specific-gravity. Hence a specific-gravity analysis unaccompanied by analytical data on the products is a botchery. As Mr. Vissac points out, specific gravity is a means, not an end.

Mr. Vissac's contributions to the concept of regarding gravity separation as a probability function are of great value, but the authors cannot agree that the position of the distribution curve at specific gravities of 1.3 and 2.2 is of no practical value. With most of the bituminous coals washed in this country, the fraction lighter than 1.30 sp gr is not only the largest single density fraction but also the most valuable one. Hence the recovery effected in this density fraction is of utmost importance rather than of no practical interest. Similarly, the biggest portion of the impurity to be removed has an average specific gravity of about 2.2; how can the percentage of this material that is improperly included in the clean coal be termed of no practical interest? In fact, it may be feared that one of the hazards involved in using error area or probable error to represent a distribution curve is that this practice tends to distract attention from the curve itself. The entire curve is important, and the entire curve is needed to show the complete picture of the separation; it cannot be represented completely—and often not adequately—by one or two parameters.

Messrs. Grounds and Needham have raised several interesting questions on the role played by particle size which unfortunately we do not have sufficient data to answer completely, although Bird[18] has discussed qualitatively the importance of the finer sizes of coal in jig operations.

Mr. Tromp's treatment of some of the data contained in the paper is convincing, and doubtless no one would question the qualitative statement that sharpness of separation is inversely related to throughput. However, as stated in the paper, the published performance data examined by the authors do not show any discernible relation between sharpness of separation and density of separation. The difference in opinion on this critical point stresses the undeniable need for a great many more performance data. In fact, without more data it is difficult to judge the merits and limitations of some of the performance criteria. From this standpoint, universal adoption in the near future of the standardized performance criteria advocated by Cerchar would appear to be premature.

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[17] R. E. Zimmerman: The Cleaning of Fine Sizes of Bituminous Coal by Concentrating Tables. *Trans.* AIME (1950) 190, p. 956; *Mining Engineering* (September 1950). TP 2875F.

[18] B. M. Bird: Chapter 22, *Coal Preparation.* Second Edition (1950) AIME.

High-Speed Classification and Desliming with the Liquid-Solid Cyclone

by Donald Dahlstrom

DISCUSSION

H. E. Criner *(Heyl and Patterson, Pittsburgh)*—Mr. Dahlstrom's paper is an interesting study on the use of a cyclone as a classifier. At the risk of some repetition I would like to review some material in a paper to which he refers. Although the Heyl and Patterson Co. has been primarily interested in the cyclone as an initial dewatering device a considerable amount of work has been done on the classification problem. During the study of a western Kentucky coal it was discovered that the particle sizes greater than 30 microns contained an average ash of less than 10 pct. The 20 to 30 micron material averaged about 12 pct ash, the 10 to 20 about 18 pct and around 2½ microns the ash approached 80 pct. Obviously if the +30 micron material could be separated from the remainder a desirable product, formerly wasted to the sludge pond, would result. A thickener capable of classifying at about 25 microns was chosen for this work. The thickener feed was 800 gpm at 10 pct concentration and it contained about 21 tons per hr of solids. The feed ash content averaged 15.7 pct. The material was recovered at 52 pct concentration and it contained 8 pct ash. Sixty four pct of the feed solids was recovered at the underflow. The overflow concentration was 4 pct and the average ash of the rejected material 30 pct.

Where we have been able to design the complete water circuit of a cleaning plant, the cyclones have usually been staged to produce a clean course product and a fine refuse. We have experimented with the use of extra classifying water which has the effect of making the flow ratio zero. Our classifying water was introduced to a separate smaller cyclone whose overflow was attached directly to the underflow of the larger primary cone. It was found possible to remove *all* fines below an arbitrarily chosen size by this method.

One means of steepening the classification curve, that is, to reject more material below the classification size and retain more material above that size is to successively dilute and reclassify the underflows by underflow staging. This method has the effect of squaring the cyclone recovery curve and therefore makes the overall recovery pass more steeply through the classification point. One such study was made on an anthracite slurry where it was desired to remove the —200 mesh material. The slurry solids contained 67 pct —200 mesh. The recovered material at the underflow of the second stage contained only 8 pct of the original —200 mesh and 86 pct of the +200 mesh contained in the feed.

The Cleaning of Fine Sizes of Bituminous Coals by Concentrating Tables

by R. E. Zimmerman

DISCUSSION

O. R. Lyons *(Republic Steel Corp., Cleveland)*—Some tests using wet tables conducted by an operating company to determine the effect of table-water density on the quality of the washed coal should be of interest. The only variable during the testing period was the percentage of solids in the table water.

Table operating data are shown in Table XV.

The ash content of the washed coal remained constant or nearly so, as the percentage of solids varied while the sulphur content of the washed coal decreased as the percentage of solids increased.

Table XV. Table Operating Data

Test No.	Average Solids in Table Water Wt, Pct	Washed Coal Ash, Pct	Washed Coal Sulphur, Pct
1	2.2	6.5	1.30
2	6.4	6.3	1.25
3	10.0	6.5	1.10

Application of Geology to Mining at Giant Yellowknife

by J. D. Bateman

At Giant Yellowknife, where high grade gold-bearing orebodies are highly irregular in shape, geology has been applied extensively to the mining of ore. The classical functions of the mine geologist in the fields of exploration and mine development have been extended to guide ore extraction, ensuring "clean" mining and effectively reducing waste dilution.

THE property of Giant Yellowknife Gold Mines Ltd. is situated west of Yellowknife Bay on the north shore of Great Slave Lake, a distance of 600 air miles north of Edmonton, Alberta.

The Giant claims were staked in 1935, the company was formed in 1937, and the main orebody system was disclosed by diamond drilling in 1944 following a geological study of the property by A. S. Dadson,* Consulting Geologist for the company. Production began in 1948 at the rate of 200 tons per day and, during 1950, reached a daily rate of 425 tons. During the first 3 years of operation a total of 366,000 tons was milled with an average grade of 0.79 oz of gold per ton. Work is in progress with an expansion to 700 tons per day in view.

A descriptive account of the geology and gold-bearing shear zone has appeared previously.*

The rock formations in the vicinity of Yellowknife Bay have been subjected to protracted pre-Cambrian tectonic deformation culminating in a series of late faults having a cumulative horizontal displacement exceeding 11 miles. The Giant property is underlain by part of an Archean sequence, several miles thick, consisting of basic volcanic flows and minor intercalated tuffs. The volcanic succession forms the west limb of a major syncline, the flows facing east, but overturned on Giant property to dip west at 65° to 75°.

Orebodies are confined to shear zones up to 200 ft in width, which were formed along early thrust faults. The shear zones assume fold-like attitudes, the larger of which have an amplitude of several hundred feet. The rock formations beyond the limits of the zone of shearing do not reflect the simulated folds, the axes of which are within a few degrees of the strike of the flows.

The schistosity and most of the planar elements in both the shear zones and the orebodies dip west at angles between 65° and 75°, generally corresponding to the dip of the flows. The planar elements within the shear zone system thus dip more or less constantly west whether the shear zone is flat, vertical, or expressed as east or west dipping limbs.

The shear zones reflect the deformation and alteration of the basic volcanic flows into chlorite schists which, in most places, have undergone metasomatic replacement to form chlorite-sericite-carbonate schists or sericite schists. The boundaries between the shear zones and country rock, although often gradational, usually can be defined within a few feet or even inches as they are expressed by the limits of metasomatic alteration.

Orebodies may occupy a small or large proportion of the shear zone and, although they generally conform to the shape of the zone, their morphology is much more complex. Ore boundaries in some instances are sharp and can be delineated with a chalk line; but more generally, a large proportion of the ore boundaries is not visually obvious and can be determined only by the perception acquired by the geological mapping of ore or study of drill cores. Ore shoots in fold-like attitudes may transect the planar elements of the shear zone at any angle; yet the schistosity within the ore shoot may be coincident with that in the enclosing shear zone. Thus it is clear that problems may arise in the delineation of mining boundaries.

Ore generally consists of 20 pct or more quartz with ferruginous carbonates in sericite schist deposited in two dominant stages. The earlier stage consists of quartz with carbonate, pyrite, and very fine-grained arsenopyrite in lenses and bands. The later stage consists of quartz-carbonate lodes, in

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* A. S. Dadson and J. D. Bateman: Structural Geology of Canadian Ore Deposits, Can. Inst. Min. Met. Jubilee Volume (1948), pp. 273-283.

J. D. BATEMAN, Member AIME, is Chief Geologist, Giant Yellowknife Gold Mines, Ltd., Yellowknife, Northwest Territories, Canada.

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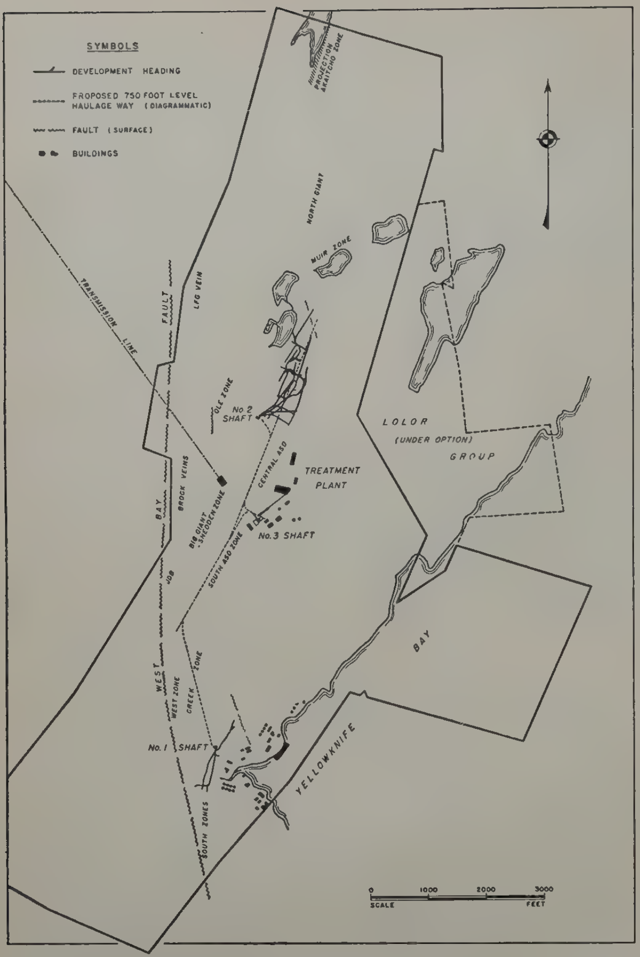

Fig. 1—Plan of Giant property showing relation of shafts and development workings to ore zones.

many cases transecting the earlier stage and distinguished by the presence of sphalerite and the sulphosalt mineral group. Gold is associated with arsenopyrite in the earlier stage, but is more abundant in the later stage of mineralization, in which fine visible gold is sometimes observed. There is a wide variety of ore types, depending upon various proportions of ore and gangue minerals. The sulphide content of an ore shoot may range from 2 to 7 pct. Orebodies are dislocated by postore faults, the largest of which, the West Bay fault, has a horizontal component of movement ranging from 15,000 to more than 25,000 ft.

There is a broad structural control of orebody distribution in the shear zone system with orebodies developed in structural anomalies of the system expressed as steeps or flats, and particularly, where the zone is sufficiently deformed to effect reverse dipping limbs. Statistically, more ore is found within or near crests than troughs within the shear zone system.

There is, further, a control of ore shoots exerted by lithology with ore mineralization showing a preference for some members of the volcanic formations over others. The length, depth, and pitch of individual shoots therefore is controlled generally by the trace of the intersection of the shear zone with the flows. Ore shoots are from 200 to 2000 or more ft in pitch length, ranging in width up to 100 ft.

Exploration

Exploration on the property is conducted by surface diamond drilling, which is planned and executed by the consulting geologist and the chief geologist. The locations and depths of holes are determined by the statistical probabilities of ore shoot distribution after assessing all geological considerations. Although diamond drilling programs are planned well in advance of the commencement of drilling, the location of each drillhole often is determined by the results of the immediately preceding hole. This flexibility is considered essential for efficient use of the diamond drill as an exploratory tool. A total of 130,000 ft of surface diamond drilling has been completed to date, and this figure is being increased by annual projects of 10,000 to 15,000 ft.

Exploratory surface diamond drilling merges with development drilling as ore disclosures are probed further in detail to arrive at preliminary estimates of tonnage and grade and to provide adequate information for the guidance of primary underground development headings.

Development

Lateral development has been undertaken from two shafts 1 mile apart; in addition, an intermediate production shaft, No. 3, was completed in 1951. The relations of the shafts to the ore zones on the property are shown in Fig. 1. Current development plans call for interconnected workings from the three shafts on the bottom or 750-ft level, which will be extended over a distance of 2 miles. Ultimately an additional mile of workings serviced by a fourth shaft will be required for the development of the northern part of the property. Limited probing has shown good ore-bearing conditions to a depth of 1250 ft, and there is no known geological limit imposed on the depth to which ore will be found.

Primary development headings from each shaft are planned by the geological department, and such headings are carried above or below, within or beside ore shoots.

The exact locations of headings with respect to ore shoots are determined by the most suitable approach for detailed underground diamond drill definition of ore, and also by the ultimate utility of the heading for the mining of ore.

As no two ore shoots are similar, the development approach to ore on each level presents special problems involving the interpretation of geological data. In the development of shoots pitching at a low angle, crosscuts commonly are driven at the intersection of the bottom of the shoot with the level, requiring critical projections of geological information. In addition to driving in ore, drifts are carried beneath an ore shoot or above the apex of a shoot, the location of the heading being determined by the projection of diamond drill data. With few exceptions all development headings are driven on predetermined lines; but in practice, diamond drilling may be done at intervals of 100 ft, and the line drive adjusted accordingly as required.

Underground Diamond Drilling

Underground diamond drilling is planned and executed by the geological staff. All development drilling is done on standard section lines at 50-ft intervals with a sufficient number of holes to provide a reasonable outline of the ore shoot. Fig. 2 is a typical development section illustrating cross-sectional diamond drilling of an ore shoot. It will be noted that the drift is simply a small opening in the ore shoot providing access for diamond drilling and, later, for a stope raise to be driven from the level below. Where the disposition of the ore is more than usually complex, cross-sectional drilling is carried out at 25-ft intervals. Following completion of development diamond drilling of an ore shoot, the ore is reclassified from indicated to developed reserves providing there are accessible openings below. A total of 150,000 ft of underground diamond drilling has been completed to date. Approximately 10 pct of this drilling is for exploratory purposes, 50 pct is classified as development drilling, and about 40 pct is required for detailed preparation of stope layouts.

Mining System Selection

Because of the varied disposition of ore, each shoot presents a specific problem of extraction. The mining system is determined, therefore, not so much by considerations of ground support as by the attitude and size of the ore shoot. All present underground workings are at relatively shallow depths, and there are no mining problems involving support of the superincumbent load. As much as 10,000 sq ft of backs have been exposed in stopes with good support throughout. Some upper level stopes are in permanently frozen ground and there are some technical problems encountered in the permafrost zone, particularly with relation to the extraction of surface pillars.

Shrinkage stoping is indicated wherever the ore can be drawn at angles steeper than the angle of rest for the broken material. In low angle or complex ore shoots, open stoping is carried out providing the height of exposed backs is maintained within reasonable limits. Cut-and-fill stoping is being initiated for the larger ore shoots with low angle attitudes. Individual stopes at No. 2 Shaft, the current source of production, range from 10,000 tons to 150,000 tons of ore, grading from 0.40 oz per ton in gold to more than 1 oz per ton. With further increases in milling rate, lower grade orebodies will be brought into production from other shafts.

Diamond drilling of ore shoots from development headings is accomplished in sufficient detail that contour plans of the ore outlines can be drawn at 10 to 20-ft vertical intervals as required. Contour maps assembling all information pertinent to the extraction of the ore shoot are prepared by the geological staff and forwarded to the engineering staff for consideration of the mining method. A mining layout is then designed by the engineering department in consultation with the mine superintendent and approved by the mine supervisory staff and the chief geologist. Fig. 3 is a typical stope preparation layout showing the positions of draw points with respect to ore boundary contours.

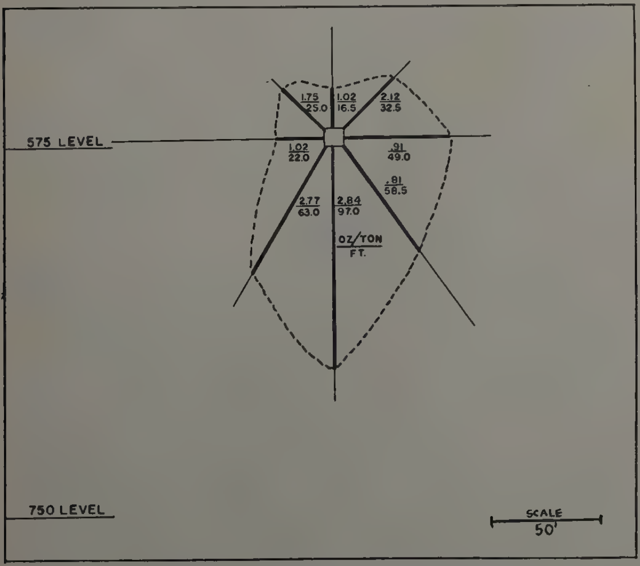

Fig. 2—Cross-section showing diamond drill definition of ore. Giant No. 2 shaft, section 1800N.

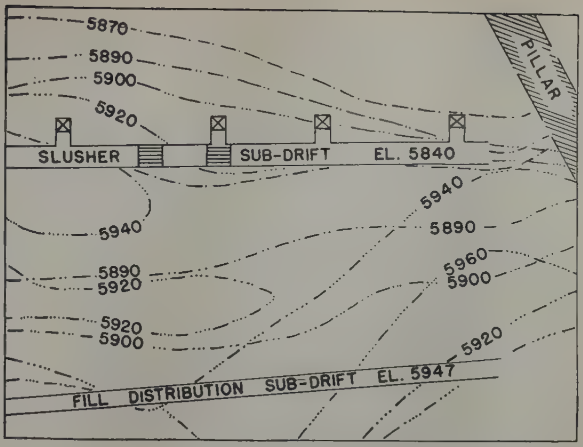

Fig. 3—Plan showing stope development layout superimposed on ore outline contours.

Mining Boundaries Control

With the commencement of preparation of a stope, geological mapping is started and is carried on throughout the life of the stope. The primary purpose of this mapping is to provide guidance for the mining of ore. All pertinent geological, assay, and diamond drill information is incorporated in 20-scale plans carried by the stope geologists. Test holes are laid out regularly by the geologists and plotted on the plans. Geologists mark with chalk all ore boundaries exposed in the stope each day and indicate the extent to which ore may extend into the walls of the stope as determined from test holes and other data. Fig. 4 is a typical cross-section through four stopes; and in examining this section, which faces north, it should be kept in mind that the lineation both in the ore and within the shear zone dips steeply west (to the left) at 65° to 75° regardless of the dip of the ore. Access to the lower stope, No. 314, is by a sublevel driven above the 425-ft level and from which the ore has been outlined by diamond drilling. The pendant of ore below the sublevel is being removed by shrinkage stoping, the broken ore being drawn on the 425-ft level. Above the sublevel the footwall of the ore is too flat for shrinkage mining, and plans are under way to remove this section up to the 250-ft level by cut-and-fill. The section above the 250-ft level was mined in 204 stope by open stoping as the ore is relatively narrow and of limited height above the level. The irregular section of ore to the east was mined in 218 stope by open stoping and movement of the ore to the draw point with a scraper. In this instance it was essential that the draw point be placed at the lowest part of the ore. The steep ore section in 212 stope was mined by shrinkage stoping and the ore removed by mucking machine crosscuts, one of which is illustrated in the section.

It is clear that no standard procedure can be used in the extraction of ore and that considerable detailed information is required preparatory to an attack on an ore shoot. It is the responsibility of the geological department to provide this information.

Dilution

When a mining plant is operating at capacity or at a fixed rate, each ton of waste that is broken in a stope and ultimately finds its way to the ore pass displaces a ton of ore in the mill and is directly reflected in production figures. The control of dilution is therefore an important factor in the economics of mining. The primary control is exercised by the most efficient mining system for the clean extraction of ore compatible with the cost of mining. The secondary control is exerted through the chalk-line definition of ore boundaries by the stope geologists, see Fig. 5.

In the mining of very irregular ore, a certain amount of dilution is to be expected no matter what mining system is employed; but if the general shape and irregularities of ore are made reasonably clear to the mine supervisory staff and to the miners themselves, inherent dilution can be greatly minimized.

The mine geologist is able to assess the quality of broken material, knowing its source, and is therefore able to distinguish broken ore from broken waste. For this purpose the geological department maintains a blackboard at each ore and waste pass listing all working places; on it broken material arriving in trains is assigned as ore or waste. In the preparation of new stopes, waste work is completed as far as possible before entries are made into ore to avoid mixing ore and waste in draw points.

Sampling

Sampling of development headings, stopes, and of broken material trammed from draw points is carried out under the direction of the geological department. Areal coverage of freshly broken faces by chip sampling gives a more reliable determination of ore grades than cutting channel samples. Current and cumulative sample averages are maintained for each working place. The primary purpose of mine sampling is to determine the grade of material broken and trammed, the broken ore being sampled from cars. The secondary purpose is to assist in the visual selection of mining boundaries. With few exceptions there are no assay walls to orebodies. Sufficient experience has not yet been gained to determine a correction factor for stope sampling. The cumulative car sample average for the first 390,000 tons milled is 0.791 oz in gold per ton as compared with a calculated mill head of 0.794 oz per ton, a very satisfactory check.

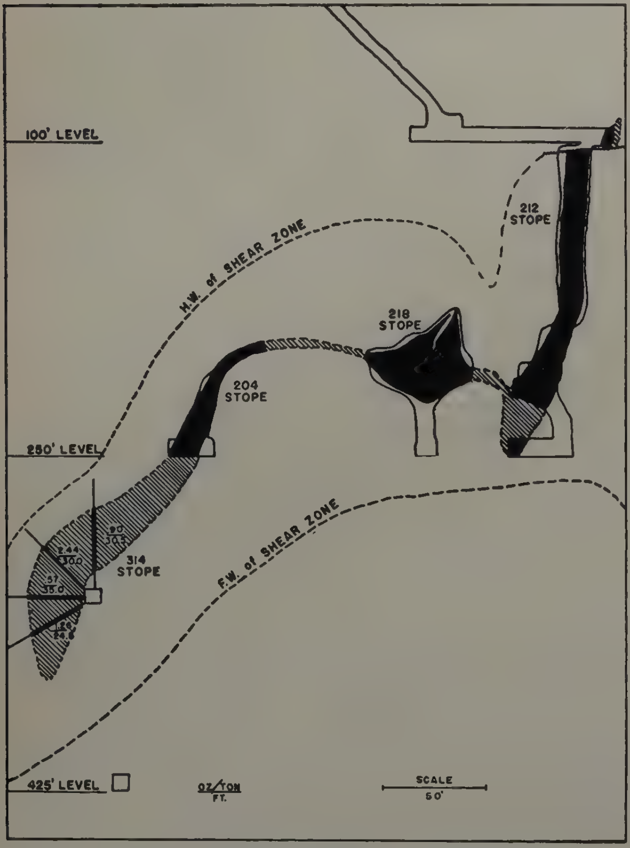

Fig. 4—Vertical cross-section through four stopes. Giant No. 2 shaft, section 850N. Stoped ore, solid black.

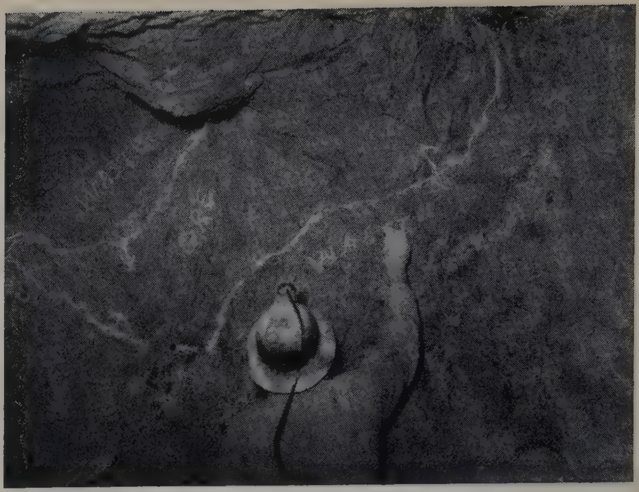

Fig. 5—Chalk line definition of ore.

Conclusions

The effective use of geology in all stages of the underground mining operation at Giant Yellowknife requires a close liaison between the geological and mine supervisory staffs. Geologists supply the mine staff with stope plans (and sections where necessary) which are compilations of geological data, incorporating significant assay information, classifying ore, and showing test holes. Such plans usually include ore boundary contours for 10 to 20 ft above current working faces, as their main purpose is not so much to record past events in the operation of a stope as to assist in predicting what is about to happen. Meetings are held twice weekly by the geological, engineering, and mining departments at which all current problems are discussed.

The position of the geologist in the control of mining boundaries is essentially advisory, involving responsibility without authority. On the basis of present experience one stope geologist is required for each 300 tons milled daily.

It is difficult to estimate in actual tonnage the degree of clean mining and control of dilution exerted by geological supervision; but it is considered that dilution is reduced by at least 6 pct and that an equivalent tonnage of ore is made available that otherwise would be missed in daily mining operations.

Acknowledgment

In conclusion, the writer would like to express his indebtedness to the late A. K. Muir, General Manager, whose cooperation has made possible the use of geology in the extraction of ore at Giant, and to A. S. Dadson, consulting geologist, many of whose ideas have been incorporated in this paper.

Basic Laboratory Studies In The Unit Operation of Crushing

by J. W. Axelson, J. T. Adams, Jr., J. F. Johnson,
J. N. S. Kwong, and E. L. Piret

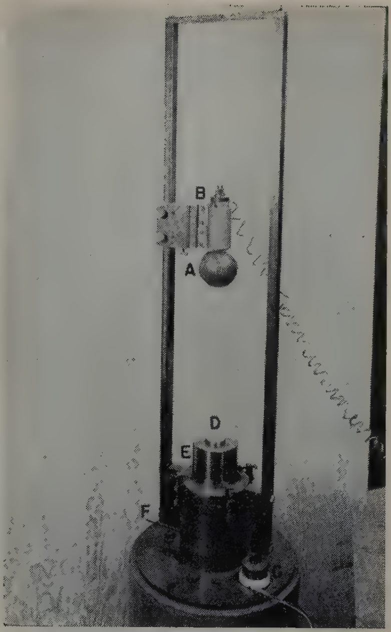

Fig. 1—Drop-weight crusher assembly.

CRUSHING has always been a major operation in the chemical and metallurgical industries, yet little is known about the theory of crushing, and today, the design of crushers is still based almost entirely on empirical knowledge and accumulated practical experience. In view of the increasing national need for the economic working of poorer grades of ores, the lack of a fundamental understanding of this unit operation hardly presents a satisfactory situation.

Basic investigations of crushing have been concerned mainly with three phases of the problem, 1—the mechanism of the fracture process itself, 2—the particle size distribution of the crushed product, and 3—the relationship between the energy input and the amount of new surface produced. Probably adequate information on these phases will be required for a comprehensive understanding of the process of crushing. A bibliography covering these phases will be given as well as a review of the work done in recent years at the University of Minnesota on the relationship between the energy input for crushing and the amount of new surface produced.

Fracture of Solids

This phase of the crushing problem is the most fundamental because it is concerned with the actual mechanism of fracture. The main problems in a study of the fracture process are concerned with the questions of why and how fracture occurs and why there is such a discrepancy between the actual and theoretical energies needed in a fracture process.

In considering the how of fracture, Poncelet[40] carried out experiments on the crushing of glass under compression. Actual photographs were taken of the glass plates in various stages of fracture. From these experiments Poncelet was able to postulate a probable mechanism of fracture.

Probably because of the quantitative nature of the problem, considerable effort has been expended to find an explanation for the low tensile strength of materials and the conversely high energy for crushing. Tensile strengths are often only 1/500 of the theoretical while the energy for crushing is usually at least 500 times the theoretical. A review of the work to explain the low strength of glass is given by Weyl,[52] in which the work of Griffith,[27] Joffé,[32] and Powell and Preston [41] are discussed. In a recent paper Seitz[46] holds that the presence of flaws and fissures, as originally expounded by Griffith, is the most probable reason for the low strength of metals.

Since a mass of data has been accumulated on the fracture strength of materials, several investigators have attempted to correlate this data in the form of an equation. Glathart and Preston,[26] Taylor,[49] Poncelet,[40] Machlin and Norwich,[37] and Frederickson and Eyring,[20] have all considered this problem, and the final relationship has always been that the stress for fracture is proportional to the logarithm of time. Murgatroyd[39] arrived at a very similar relationship by a different method.

The study of the fracture process has used many methods. Among some of these are the thermodynamic approach used by Fürth[21] which was based on the work of Born[11] and later used by Saibel,[43] and the statistical method applied by Fisher and Hollomon.[19]

In recent years metallurgists have concerned themselves with the problem of brittle fracture as opposed to the usual plastic failure of metals. The great amount of interest shown is reflected by the number of contributors to the recent symposium on the fracture of metals.[2] Repeated failures of welded ships during World War II have accelerated these studies.[34, 50, 54]

The shifting of thought from an emphasis on the

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Discussion of this paper, TP 3194B, may be sent to AIME before Jan. 31, 1952. Manuscript, Dec. 17, 1949. Revision, March 15, 1951. St. Louis Meeting, February 1951.

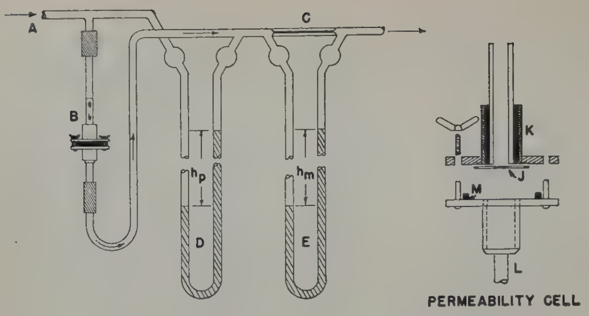

Fig. 2—Air permeability apparatus.

importance and significance of the concept of a critical stress and tensile strength measurements in the study of fracture of materials to an interest in the understanding and measurement of the energy required to produce fracture is believed by the authors to be highly desirable. This will tend to bring the work on fracture and on crushing closer together.

Size Distribution of Crushed Material

The size distribution of crushed materials has been studied for many years[10, 15, 22, 23, 29] but the information obtained was generally used as a measure of the surface area or the state of the final product and not as a means of studying the actual mechanism of crushing. Screen analyses run on crushed samples have established a definite relationship between the weight retained on a screen and the size of the screen opening. Schuhmann[44] presents some of the latest work on this. The same relationship has been used by Bond[10] in deriving his method of evaluating surface areas of crushed materials from a screen analysis.

Gaudin and his associates[24, 25] have used their data on size distribution to derive some hypotheses on crushing, making use of various postulates by Andreasen[3] and Bennett, Brown, and Crone[8] to synthesize size distribution curves which agree very well with experimental data. Epstein[17] has used an idea similar to that of Bennett, et al, in his recent study of the size distribution of a crushed solid. From the experimental data which they obtained Gaudin and his associates made the amazing conclusions that "1—The size distribution of broken fragments made by a single fracture is such that the new surface on each grade is the same; and 2—Multiple fracturing results in a size distribution such that in the fine sizes, the surface per grade is the same in every grade; in coarse sizes the surface per grade decreases gradually with increasing size." This is about the only work that attempts to make use of the size distribution of a crushed product as a key to the understanding of the fracture process.

Energy Input Vs. New Surface

Prior to 1928 a considerable portion of the literature in the field of crushing[22, 23, 31, 48] was concerned with the controversy between the supporters of Rittinger's law and the advocates of Kick's law. Even though Gross and Zimmerley[29] seemed to have demonstrated quite clearly in 1928 that Rittinger's law is valid, at least for quartz and for some conditions, the dispute over the two laws still continued.[7, 14, 52]

Energy Measurement: Studies of the crushing process have been complicated by the fact that a valid measurement of the energy input to a crushing operation is difficult to measure. Several attacks are possible. One can try to measure, for example, the energy input to a commercial machine,[6, 38] the energy input to a bed of particles in a simple crushing device,[9, 22, 24, 29, 33, 35, 42, 53] the energy input to a single particle,[5] or measure the difference in the total energy levels of the material before and after crushing. The last is the most basic but is difficult to perform today.

Surface Area Measurement: Although surface measurements made on fine materials by means of gas adsorption and the electron microscope are probably a close approach to the true value,[4, 13] these methods were not available until recently. The most common method of determining the surface area of particles has been the indirect one of obtaining a screen analysis and more or less arbitrarily assigning a shape factor to correct for the assumption that the particles were perfect spheres or cubes. The large errors that can arise from this assumption have been demonstrated by Gross and Zimmerley[29] and Gaudin and Hukki.[24] Bond[10] has developed a graphical method of determining areas from a screen analysis which results in a much improved value for the surface of the material smaller than the smallest sieve size. Martin[38] in 1925 and Gross and Zimmerley[29] calculated the surface area of crushed quartz particles by measuring their rate of dissolution in hydrofluoric acid. While this method undoubtedly results in good values, it is not easy to perform, cannot be used on heterogeneous material, and the solvent used has to be specific for the material tested. Furthermore, some of the material is dissolved during the area determination and so cannot be used for additional experiments.

Quite a number of investigators[1, 4, 15, 33, 35, 36, 45] have calculated surface areas of particles by the permeability method first introduced by Carman[15] in 1937. Although this method probably does not give a measurement of the true surface area, it is rela-

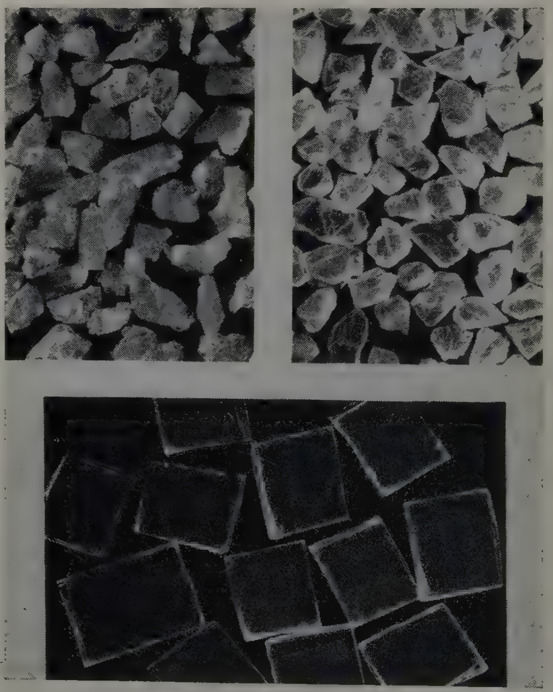

Fig. 3—Micrographs of materials crushed. Top left—14/20 mesh labradorite, X3.5. Top right—8/10 mesh fluorite, X2.3. Bottom—Glass pieces, 0.9x0.9x0.25 cm.

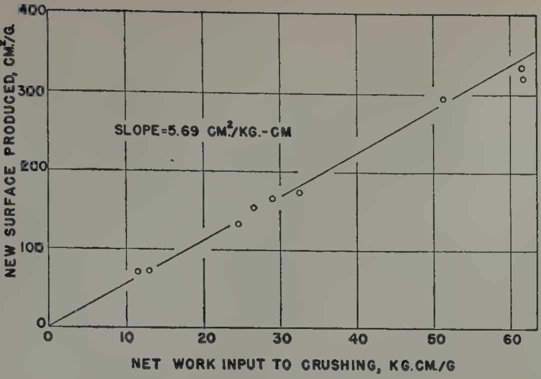

Fig. 4—Energy-new surface relation for impact crushing of milky vein quartz.

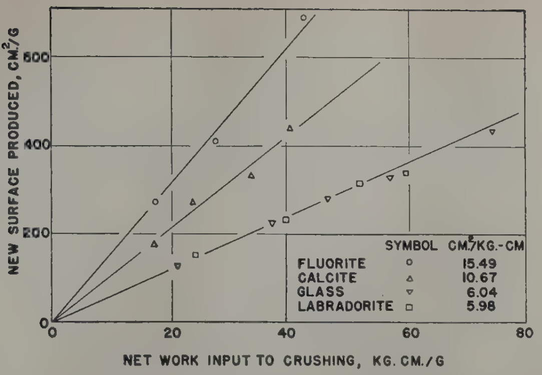

Fig. 5—Energy-new surface relation for impact crushing of several brittle materials.

tively simple and easy to use. Turbidity measurements of surface have been highly developed[47, 53] but are not applicable to the larger mesh sizes usually encountered in any crushing experiments.

The most promising and most recent method for surface area determination is the gas adsorption method. Wooten and Brown[55] have adopted the general adsorption method developed by Brunauer, Emmett, and Teller[12] to the determination of small areas, and this has considerably increased the range of areas which can be measured. This method was used by Gaudin and Hukki[24] and at the University of Minnesota[5, 33] in studies on the crushing of quartz.

During the past decade several investigators working under E. L. Piret at the University of Minnesota have conducted a fundamental study of the process of crushing. Most of this work has been concentrated on crystalline quartz, but other minerals have also been investigated. The primary interest has been the relationship between the energy input and the new surface produced in a crushing operation.

Permeability Surface Measurement: A considerable portion of the work done at the University of Minnesota[1, 33, 35] made use of a drop-weight crusher, see Fig. 1, similar to that used by Gross and Zimmerley.[29] In this apparatus the material is placed in the steel mortar *E*, and is crushed by the impact of the steel ball *A* as it drops on the cylindrical plunger *D*. To prevent rebound of the ball, three aluminum wires are spaced equally under the mortar. When the ball is dropped with no material in the mortar, the deformation of the wires is taken as a measure of the kinetic energy of the falling ball, and it is assumed that a similar wire deformation with material in the mortar represents an equal amount of energy. Therefore, in a crushing experiment, the energy utilized for crushing is the difference between the kinetic energy of the falling ball and the energy represented by the deformation of the aluminum wires. This latter energy is obtained from the calibration curve of wire thickness vs. energy input.

Fig. 2 is a diagrammatic sketch of the air permeability apparatus used to measure surface areas in the initial work. The air enters at *A*, flows through the bed of material *B*, through the flowmeter *C*, and finally to a vacuum pump. The pressure drop, hp, across the bed of material is measured by the water manometer *D*, and the rate of air flow is measured by the pressure drop h_m, across 300 cm of 1 mm ID glass capillary tubing represented by *C*. If these pressure drops and the thickness and porosity of the bed are known, the surface area of the particles can be calculated easily. In some of the measurements on coarse material, a water permeability apparatus of a similar type was used.

Fig. 3 is a photograph of some of the materials crushed. The samples were always in the size ranges shown close to them.

Table I gives a summary of the results of a series of crushing experiments with a variety of minerals using sample weights of 12 to 50 g. Figs. 4 and 5 are plots of the relationship between the new surface produced and the net work input for most of these materials. The slopes of the lines in these plots indicate the amount of new surface formed per unit of energy input, and this value is seen to decrease from fluorite through calcite, glass, labradorite, and quartz. These same values are given in col. 6 of Table I for all the materials, and Fig. 6 is a plot of these values vs. the Mohs' hardness of the materials. Although a few materials do not agree too well, the plot does show that the amount of new surface formed per unit of energy input decreases as Mohs' hardness increases and is essentially a straight line relationship.

All of the materials considered so far have been supposedly brittle materials. The crushing of a nonbrittle but crystalline solid is now considered. When sodium chloride was crushed as described in this section,[1] the relationship between the new surface and the net energy input was found to be a curved line as shown in Fig. 7. Since sodium chloride has considerable plasticity under certain conditions, plastic deformation was suspected as the reason for this unusual behavior. X-ray photographs of the crushed salt showed a definite asterism of the Laue spots which were not present in the uncrushed material or in the crushed brittle solids. This asterism is taken as evidence that plastic deformation did occur. Since no part of the energy causing plastic deformation appears as new surface and the amount

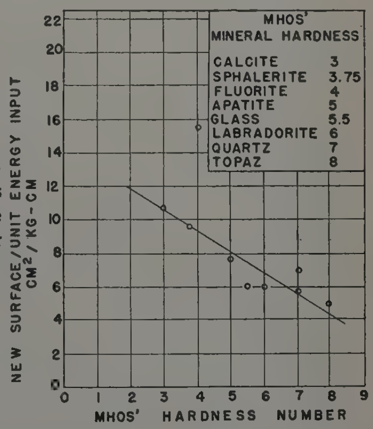

Fig. 6 — Relationship between the Mohs' hardness of various minerals and the surface produced per unit of energy input.

Table I. Summary of Results Using Drop-Weight Crusher and Air Permeability

Material and Experiment No.	Net Work to Crushing Kg-Cm per G	Initial Surface of Sample, Sq Cm per G	Surface of Crushed Product, Sq Cm per G	New Surface Produced in Crushing, Sq Cm per G	Surface Produced Per Unit of Work, Sq Cm per Kg Cm	Crushing Resistance, Kg Cm per Sq Cm
Quartz						
(Milky vein)						
F	12.8	127.8	202.3	74.5	5.81	0.172
G	11.54	127.8	203.0	75.2	6.51	0.154
H	29.1	43.1	210.6	167.5	5.76	0.173
I	26.5	43.1	198.5	155.4	5.87	0.170
J	62.1	14.1	335.9	294.2	5.18	0.193
K	51.6	43.1	337.3	321.8	5.71	0.175
L	61.8	14.1	348.0	333.9	5.40	0.185
M	24.4	43.1	179.5	136.4	5.60	0.179
N	32.5	74.1	249.4	175.3	5.40	0.185
					Avg 5.69	0.176
(Crystalline)						
Cr-1	7.97	1.45	63.75	62.3	7.81	0.128
Cr-2	37.14	40.7	266.0	225.3	6.06	0.165
					Avg 6.94	0.147
Calcite						
C-1	23.3	23.1	296.0	272.9	11.70	0.0855
C-2	17.59	23.1	200.7	177.6	10.10	0.0895
C-3	40.06	23.1	470.5	447.4	11.17	0.0900
C-4	33.5	58.3	383.5	325.2	9.72	0.103
					Avg 10.67	0.092
Fluorite						
F-1	41.92	14	697.0	683.0	16.3	0.0613
F-2	28.1	14	421.2	407.2	14.5	0.0690
F-3	17.37	14	496.2	272.2	15.67	0.0638
					Avg 15.49	0.0647
Labradorite						
L-1	23.63	43	192.0	149.0	6.31	0.159
L-2	38.55	43	274.3	231.3	6.00	0.167
L-3	51.81	43	351.5	308.5	5.95	0.168
L-4	58.78	43	376.0	333.0	5.67	0.177
					Avg 5.98	0.168
Sphalerite						
S-1	26.97	6.2	278.3	272.1	10.08	0.099
S-2	50.54	6.2	463.5	457.3	9.06	0.110
					Avg 9.57	0.105
Apatite						
A-1	41.18	70.7	389.3	318.6	7.74	0.129
A-2	55.6	70.7	492.2	421.5	7.58	0.132
					Avg 7.66	0.1307
Topaz						
T-1	53.3	1.1	268.2	267.1	5.00	0.200
Glass						
G-1	20.85	5.0	141.1	136.1	6.58	0.153
G-2	36.75	5.0	131.6	226.6	6.16	0.163
G-3	56.83	5.0	329.0	319.0	5.61	0.178
G-4	74.17	5.0	436.0	431.0	5.81	0.172
G-5	46.70	5.0	287.0	282.0	6.06	0.165
					Avg 6.05	0.166

Table II. Summary of Results Using Drop-Weight Crusher and Gas Adsorption

Material and Experiment No.	Net Work to Crushing, Kg Cm per G	Gas Adsorption Data			Air Permeability Data			Ratio of Final Surfaces, Adsorption to Permeability
		Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm Per G	Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm Per G	
Crystalline quartz								
Q-1	24.4	1350	8130	313	635	3960	153.5	2.05
Q-3	28.3	1080	6830	340	516	3070	155.0	2.22
Q-4	27.0	1130	6550	323	522	3450	174.2	1.90
Q-5	14.1	435	1230	158.8				
Q-8	134.1	385	6250	1170	155	3320	636	1.88
Q-9	191.4	404	8370	1590	155	4410	848	1.90
Q-10	37.1	364	2620	450				
Q-11	23.5	370	1940	315				
Q-12	291.0	370	11990	2320	155	6250	1220	1.92
Q-13	58.3	372	3800	683	155	2150	398	1.77
Q-14	22.0	387	1820	287				
Q-15	23.6	710	3750	305				
Q-16	21.4	3220	4770	309				
Q-17	21.1	1410	2750	268				
Q-18	98.2	3200	8140	985				
Massive quartz								
HQ-1	76.8	656	4630	794	220	2640	484	1.76
HQ-2	154.5	680	7930	1450	220	4230	800	1.87
Milky vein quartz								
MQ-1	95.5	1075	6300	1100	150	3200	610	1.97
MQ-2	53.2	1075	4330	660	150	2034	377	2.13

Table III. Summary Data Using Slow Compression Crusher and Gas Adsorption

Experiment No.	Net Work to Crushing, Kg Cm per G	Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm per G
CQ-12	177.0	370	12270	2350
CQ-13	158.0	340	9870	2100
CQ-14	107.8	370	8105	1550
CQ-15	65.9	370	5200	965
CQ-16	30.6	370	2235	370
CQ-17	8.9	330	1035	140
CQ-18	131.9	370	10000	1925
CQ-19	170.6	390	12940	2500

of this energy is proportional to the net energy input, it appears logical to assume that plastic deformation caused the curvature in the relationship between new surface and energy input.

Adsorption Area Measurement: Since there was always some doubt as to what surface was being measured by the permeability method, it was decided to build a gas adsorption apparatus to use for measuring surface areas, see Fig. 8. *A* and *B* are the diffusion and the mechanical pumps, *C* and *D* are McLeod gages, *E* is the sample tube, *F* is a vapor pressure bulb, *G* is a gas reservoir, and *H* is a cold trap. The nos. 1 through 5 designate mercury cutoffs used to isolate parts of the system. A complete description of this apparatus and its use is given in reference 33. Suffice it to say that ethane was the adsorbent at the temperature of liquid oxygen and that adsorption was carried out in the pressure range of 0.5 to 3.5 microns. The Brunauer, Emmett, Teller method was used to calculate surface areas.

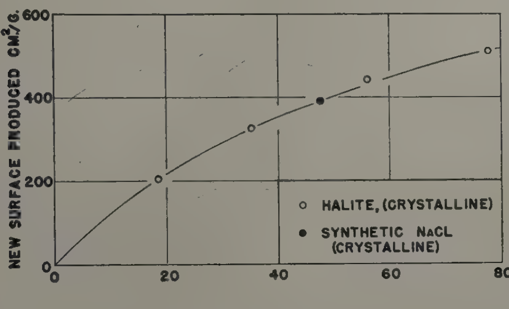

Fig. 7—Energy-new surface relation for impact crushing of salt.

A series of experiments was conducted with 5 to 20-g samples of quartz in which the crushing was accomplished with the drop-weight crusher and the areas were measured by the gas adsorption method and, in some cases, also by the air permeability method. The results of this work are tabulated in Table II and plotted in Fig. 9. Although Fig. 9 shows a curved relationship between the new surface and the net energy input instead of the straight line relationship previously obtained, there is no discrepancy since the previous work covered such a small range that the curvature was not evident. Fig. 10 is a plot of the areas of some samples as measured by gas adsorption vs. the areas of the same samples as determined by air permeability. It can be seen that the ratio of adsorption area to permeability area is fairly constant over the range studied and is equal to about 1.9. In some later work with single quartz particles with ground surfaces, the ratio of adsorption area to geometric area was as large as 17.

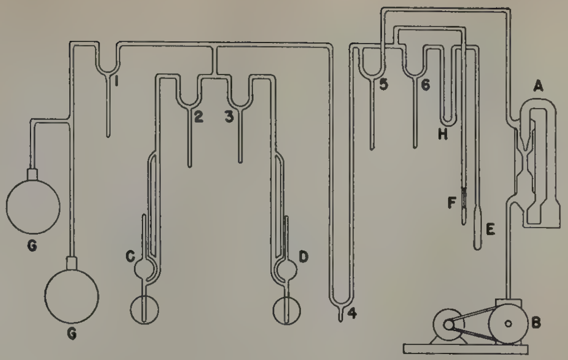

Fig. 8—Gas adsorption apparatus.

Slow Compression Crusher: In an attempt to obtain a more accurate measure of the actual energy input to the material, a series of experiments was conducted[33] in which the crushing of quartz was accomplished by slow compression in a mortar between the platens of a small hydraulic press. Fig. 11 shows the crushing assembly used. The material is contained in a steel mortar *A* which is placed on the ram of the press as shown. Two dial gages *B* attached to a steel cylinder between the top of the mortar plunger and the top platen of the press serve to measure the displacement of the applied force. Since the energy imparted to an empty cylinder with the application of a load was found to be elastically returned with the release of the load, the net displacement noted during a crushing experiment represented energy imparted to the material being crushed. The actual energy imparted to the material was computed from the area under the curve obtained by plotting displacement vs. applied force. A typical plot is shown in Fig. 12.

In the first series of experiments using slow compression crushing, 5-g samples of 10/14 mesh crystalline quartz were used, and the areas were measured by the gas adsorption technique previously described. A summary of the data is given in Table III, and Fig. 13 is a plot of the new surface produced vs. the net energy input. A straight line relationship was again established. The discrepancy between this work and the previous work with the drop-weight crusher will be explained later.

Crushing of Single Particles of Quartz: All of the work with various minerals and the different methods of crushing, energy determination, and area measurement, accumulated valuable data but did not seem to give any explanation as to the basic mechanism of crushing. Therefore, a series of experiments was conducted in which single particles of crystalline quartz were crushed by the slow compression

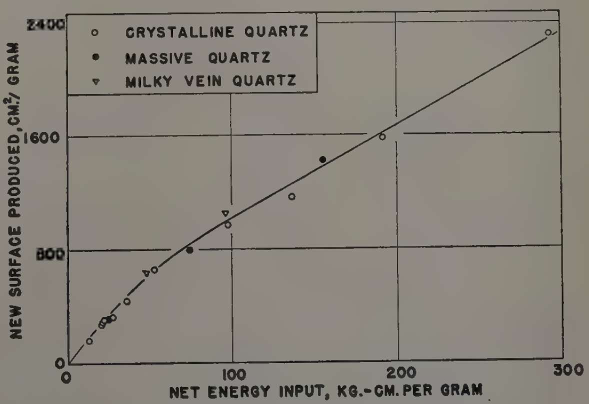

Fig. 9—Energy-new surface relationship for impact crushing of quartz. Surfaces measured by gas adsorption.

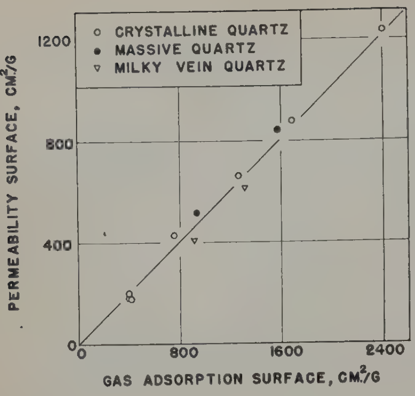

Fig. 10—Plot of surface measured by air permeability vs. surface measured by gas adsorption. Crushed quartz samples.

method, and the areas were measured by gas adsorption.[5] The quartz specimens usually weighed between 1 and 2 g. In some experiments a natural crystal was used while in others the crystal was ground to a cubical shape, but the crushing force was always applied to two $10\bar{1}0$ faces of the crystal. When this material was subjected to the compressive force of the press, a definite displacement could be measured up to the point of fracture. Some cracking or spalling of the specimen was audible during the force application, and the final crushing usually occurred with explosive violence, including a sharp report and the shattering of the original particle into very small pieces as shown in Fig. 14.

Since the return of the elastic energy imparted to the crushing assembly during compression could not be measured after fracture occurred as with the multiple particles, a correction was made for this. Therefore, the net energy for crushing was taken as the area under the force-displacement curve where the displacement is the measured displacement minus the calibrated displacement of the crusher assembly. Fig. 15 shows the general nature of these curves.

Table IV gives a summary of the data for the single particle experiments and Figs. 16 and 17 are plots of these data as the new surface formed per unit of energy input vs. the average energy concentration. For comparison, the plots also show the previous results obtained with multiple particles. It has been pointed out to the authors that an equation for the data in Table IV and the curve in Fig. 16 can be written in the following form:

$$S_u = 189/E_c^{0.689}$$

where S_u is the new surface produced per unit of energy input in sq cm per kg-cm and E_c is the energy concentration in kg-cm per g.

If the theoretically calculated surface energy of a material based on interatomic forces is taken as the criterion of the energy necessary to produce new surface, the energy used in crushing multiple particles in the drop-weight apparatus or the slow-compression assembly is found to be about 50 to 100 times the theoretical amount. Therefore, if these surface energy figures are accepted, either present day methods of crushing are very inefficient in the utilization of the energy imparted to the material or the crushing process is inherently an inefficient process.

The results described for single particles of quartz indicate that the efficiency of the crushing process can, in some cases, be as much as 19, and perhaps even more times, greater than the usual value obtained for the crushing of multiple particles. However, even under these conditions of apparently high efficiencies, the actual efficiency is only about 25 pct when based on a value of 980 erg per sq cm for the surface energy of quartz. This value is an arbitrary one based on several theoretical values available.[5] It would appear that a large amount of the energy in any crushing operation is perhaps necessarily lost, largely as heat, without producing any new surface. Fahrenwald, et al[18] substantiated the earlier work of Cook[16] by demonstrating the actuality of a heat loss of 75 to 94 pct in a ball mill, but the necessity of such a loss has not yet been clearly established.

With only 1 to 2 pct of the energy input for the crushing of multiple particles appearing as surface energy in the crushed product, it seems quite improbable that a constant relationship could occur over any appreciable range between the energy input and the new surface produced. The straight line relationships which have been found experimentally are believed to be the result of a statistical average for the large number of particles crushed. The work with the single particles of quartz has shown that a variation of as much as 1500 pct can be found for this relationship between any two particles, but an average value would be expected, and apparently was obtained, when 500 or more

Table IV. Summary of Data for Crushing of Single Particles of Quartz by Slow Compression

	Particle Size								
Experiment No.	Cross-Section,[a] Sq Cm	Thickness,[b] Cm	Total Energy Input, Kg Cm	Average Energy Concentration, Kg Cm Per G	Initial Surface, Sq Cm	Final Surface, Sq Cm	New Surface, Sq Cm Per G	New Surface Per Unit of Energy Input, Sq Cm Per Kg Cm	Theoretical Efficiency, Pct[d]
SQ-10	0.60	0.60	1.2	0.6	124	430	159	265	26.5
SQ-6	0.47	0.55	2.7	4.2	43(c)	537	750	180	18.0
SQ-8	0.28	1.53	0.3	0.3	64(c)	118	50	166	16.6
SQ-15	0.66	0.87	2.4	1.6	78	415	218	139	13.9
SQ-17	0.49	1.10	2.0	1.5	69(c)	307	173	115	11.5
SQ-13	0.36	0.60	2.1	1.7	102	326	180	106	10.6
SQ-16	0.36	0.90	2.0	2.3	49	243	226	98	9.8
SQ-4	0.78	1.20	28.8	11.0	70(c)	2295	849	77	7.7
SQ-21	0.47	1.15	6.8	4.8	69(c)	538	330	69	6.9
SQ-9	0.83	0.55	15.4	13.6	70(c)	1140	940	69	6.9
SQ-14	2.12	1.50	35.9	2.5	344(c)	2470	150	60	6.0
SQ-20	0.43	1.04	12.3	10.4	61(c)	771	603	58	5.8
SQ-12	0.85	0.65	14.5	6.1	94	884	335	55	5.5
SQ-18	0.46	1.26	7.4	4.9	74(c)	466	261	53	5.3
SQ-11	0.46	0.50	9.8	6.1	101	496	245	40	4.0
SQ-19	0.39	1.10	20.0	17.2	60(c)	604	466	27	2.7
SQ-7	0.70	0.55	43.0	44.6	55	789	761	17	1.7

[a] Average area of two $10\bar{1}0$ faces in contact with mortar bottom and plunger.
[b] Measured in direction of force application.
[c] Area calculated from geometric area and surface factor of 17.
[d] Based on value of 980 ergs per sq cm for the surface energy of quartz.

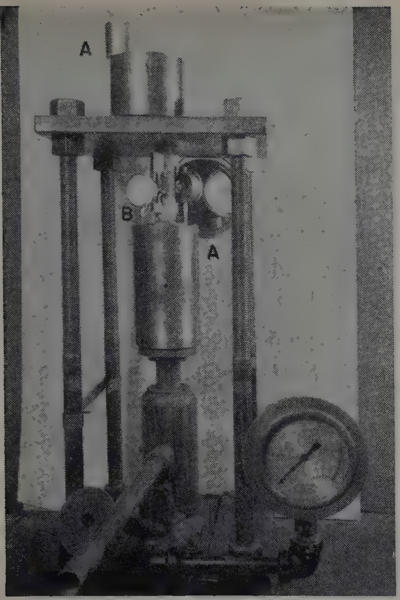

Fig. 11—Crushing assembly.
A—Mortar and plunger.
B—Dial gage assembly.

particles were used in the multiple particle experiments.

A direct comparison of the results for the drop-weight method of crushing and energy measurement, and the results obtained with the slow compression, force-displacement method can be made from the results shown in Figs. 9 and 13 and replotted together in Fig. 18. Although experimental evidence is lacking, the difference in results is attributed to a change in the usage of the energy imparted to the bed material as crushing proceeds in the drop-weight machine. In a proposed explanation it is assumed that the average stress concentration in the material with the drop-weight machine is a function of the energy input for each drop of the ball, and that a critical stress concentration is necessary before fracture will occur. With slow compression, the average stress concentration is independent of the energy input because the maximum force was kept constant. With the first drop of the ball in the drop-weight machine, the energy imparted to the bed of material is quite efficiently used to produce new surface because the average stress concentration in the bed of particles is relatively high and many particles reach the critical stress concentration necessary for fracture. With many successive drops of the ball, the energy input to the bed of particles decreases for a single drop of the ball by as much as 40 to 50 pct, and the average stress concentration is therefore less, if it is assumed that the stress concentration is a function of the energy input. As a result, fewer particles attain the critical stress concentration necessary for fracture, and more energy is lost without producing any new surface. This results in a successively lower value of new surface produced per unit of energy input as crushing proceeds in the drop-weight machine. The original work[29, 35] using the drop-weight crusher showed that the energy input per drop of the ball had no effect on the final relationship between the new surface produced and the energy input but this conclusion was based on a relatively low energy input where the effect also is not very noticable in the present work.

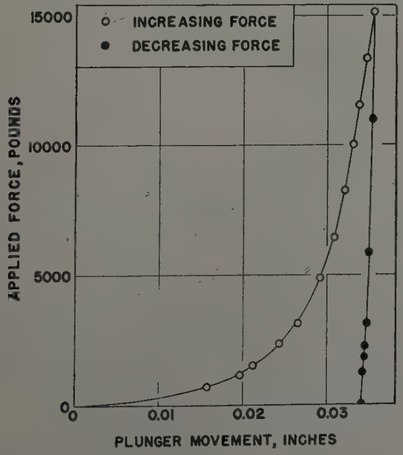

Fig. 12 — Typical plot of applied force vs. plunger movement for slow compression crushing.

Fig. 19 illustrates the differences in the areas of crushed quartz obtained by Gross and Zimmerley using the dissolution method and by the authors using the gas adsorption technique. As noted above, the curved relationship shown, curve *D*, is attributed to a change in the crushing process and not to the area measurement. Crystalline quartz was used in establishing this curved relationship except for two experiments, which were performed using a milky quartz believed to come from the same vein as that used by Gross and Zimmerley. It was found that the two types of quartz gave the same results. Since the same type of crusher and energy measurement was used in determining the relationships in curves *A*, *B*, and *D*, the results should be directly comparable. Gross and Zimmerley, curve *A*, obtained values of new surface by the dissolution method which were about 25 pct higher than those obtained by the authors in the same range using gas adsorption, curves C and *D*. This difference could be caused by a lack of consideration for the influence of the very fine material in the dissolution method as proposed by Hancock.[30] The latter has recalculated some of Gross and Zimmerley's data and obtained a value of new surface only 50 pct of that originally obtained as shown by curve *B*.

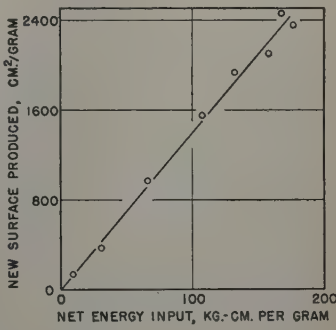

Fig. 13—Energy-new surface relationship for slow compression of crystalline quartz. Surfaces measured by gas adsorption.

It is significant that in the single crystal experiments the amount of new surface formed per unit of energy input was always greater than in the multiple particle experiments. At the lower average energy concentrations this ratio was as high as 19 to 1. Although the data for the single crystal experiments are quite scattered and group themselves most frequently around an average energy concentration near 5 kg-cm per g, a relationship is shown between the new surface formed per unit of energy input and the average energy concentration at fracture. One explanation of this relationship makes use of the concept of a critical stress value to produce fracture.

If the crushing takes place at a low average energy concentration, it is postulated that there must be a stress concentration at one or relatively few points. When fracture occurs, it originates at these few points of stress concentration, and each fracture can proceed to the edge of the crystal with a relatively

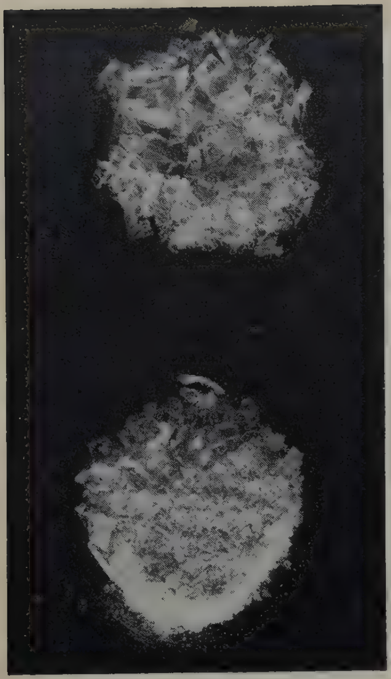

Fig. 14 — Single quartz crystals after crushing, X1.5. Top—experiment SQ-17. Bottom—experiment SQ-20.

large production of surface. Relatively few particles are formed, as shown in Fig. 14. The energy in the piece, other than at these few points of stress concentration, is lost as sound, vibration, etc. without any surface production.

When fracture occurs at a high average energy concentration, fractures are postulated to originate in rapid succession at large number of points, and a relatively large number of particles are formed, see Fig. 14. In this case, the critical stress concentration may be attained at a large number of points in rapid succession because of the high energy concentration and the irregularities or flaws on the immediately previous fracture surfaces. Now, instead of proceeding to the edge of the piece, each fracture proceeds only a short distance, with a corresponding small surface production, when it is stopped by the free surface of another fracture. Since the average energy concentration at fracture is higher than in the previous case, more energy is lost without surface production. The net result is a smaller production of new surface per unit of energy input as the average energy concentration increases because of a smaller surface production by each fracture and a greater loss of energy as a result of the higher energy concentration.

The theoretical efficiencies obtained with the experimental crushing machines described in the introduction are of the order of 1 to 2 pct. On the same basis (using a value of 980 erg per sq cm for the surface energy of quartz) the efficiencies obtained by the authors for the crushing of quartz were 1.4 pct for multiple particles and from 1.7 pct to as high as 26.5 pct for single particles, with an average of 9.4 pct for the single particle experiments performed. Since the average value obtained for the single particle experiments is about seven times the value obtained in multiple particle crushing, a potential opportunity for considerable improvement in the design and operation of crushing machines is indicated.

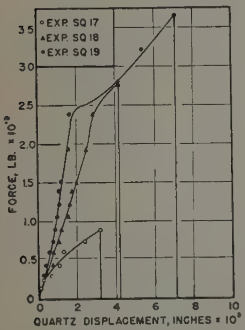

Fig. 15 — Force-displacement curves for crushing of three different quartz particles.

As a consequence of the wide variation in the performance of individual particles, whether caused by variations in size or number of flaws, imperfections of contacts, some statistical or energy jump or other effects, and despite their demonstration for many cases of straight line relationships between energy input and new surface produced for the crushing of multiple particles in a bed, the authors feel that the law proposed by Rittinger is not a basic concept in crushing and should not be used as such. It is rather the result of a statistical average effect in crushing many particles and probably need be used with caution and with regard to the conditions of the experiment and the material. The curved relation for single particles shown in Fig. 16 demonstrates a much more fundamental phenomenon of crushing. Fig. 18, in turn, illustrates the importance of the conditions of the experiment and Fig. 7 shows the importance of plastic deformation in some materials. Rittinger's law is probably too simple a concept to encompass a complicated phenomenon and may very well be a particularly desirable base or starting point for the development of a full understanding of crushing. Present studies in this laboratory are therefore pointed towards further experiments on single crystals and towards investigation of such intensive variables as temperature.

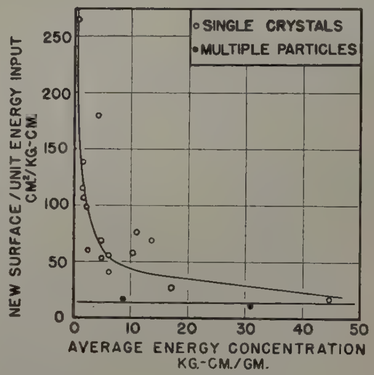

Fig. 16 — Relationship between new surface per unit of energy input and average energy concentration in crushing crystalline quartz.

Acknowledgment

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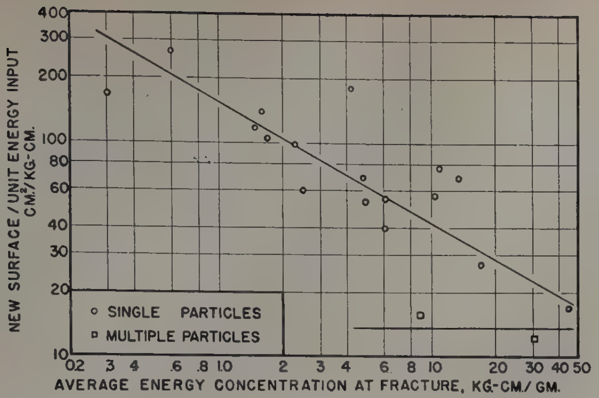

Fig. 17—Log plot of data in Fig. 16.

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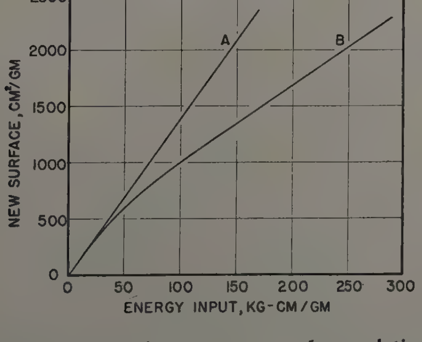

Fig. 18—Comparison of energy-new surface relationship for quartz with crushing by slow compression method and by drop-weight method.[34]
A—Slow compression, areas by gas adsorption.
B—Drop-weight, areas by gas adsorption.

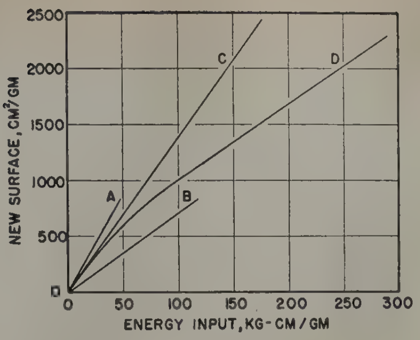

Fig. 19—Energy-surface relationship for the crushing of quartz by various investigators.
A—Drop-weight, dissolution.[30] B—Same as A, corrected by Hancock.[31]. C—Slow compression, gas adsorption.[34] D—Drop-weight, gas adsorption.[34]

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Tungsten Carbide Drilling On The Marquette Range

by A. Eugene Lillstrom

IN the development of iron mines and production of iron ore from the Marquette range, drilling blastholes is an important phase of the mining cycle. The ground drilled in ore production can be classified into two main categories, soft hematite and hard hematite or magnetite. Within these categories the material exhibits a wide range of penetrability by percussion drills.

Development work encounters various types of rock. Slate and altered basic intrusives constitute the softer types commonly encountered. Harder materials are represented mainly by greywacke, quartzite, iron formation, and diorite.

Prior to the first tungsten carbide trials in late 1947 and early 1948, hard-rock and ore drilling was done with steel jackbits starting at 2¼-in. diam. These were reconditioned by hot milling. Automatic or handcrank 3½-in. drifters were employed, mounted on Jumbos, posts and arms, or tripods, depending upon the working place. With the exception of shaft sinking jobs where 55-lb sinker machines were and still are used with 1-in. quarter octagon steel, the other production and development mining utilized 1¼-in. round and Leyner-lugged steel. The following properties have been selected as typical examples wherein carbide bit applications have proved economical. The Mather mine "A" and "B" shafts and Cleveland-Cliffs Iron Co. mines are soft ore mines where insert bits are used in rock development only. The Greenwood mine, Inland Steel Co., Champion mine, North Range Mining Co., and Cliffs shaft mine, Cleveland-Cliffs Iron Co., are hard ore mines where all drilling is done with tungsten carbide bits.

Mather Mine "A" Shaft

In the Mather mine "A" shaft and other soft ore properties where only rock development work is done with the tungsten carbide bits, several types and makes of bits have been tried since early 1948.

The greatest proportion of failures have been at the connection end, although the early trials with the 13 Series Carset 1½-in. bit used in conjunction with 3½-in. automatic-feed drifters, showed an equal amount of shattered inserts. To combat this shattering, the 3½-in. drifters were replaced by 3-in. drifters, thus eliminating, for the most part, insert failures. However, the attachment end of the rod continued to be the main source of trouble. The greatest amount of failure was in the stud or at the upset section approximately 2 in. behind the drive shoulder of the rod. Heat treatment was changed several times as well as the composition of the alloy studs. Since this failed to correct the trouble, a decision was made to change to a heavier attachment section. Timken 1½-in., type M, bits were then employed and showed an exceptional improvement. The rods are discarded when the thread contour shows sharpening or wear on the shoulder. It was also learned that the Timken insert did not show as rapid gage and cutting edge wear as did competitive makes, and footage per use increased by approximately 50 pct.

Prior to the Timken trials the average life per bit at the Mather mine "A" shaft on 6-ft change chain-feed drifters was 500 ft, and the rod life at the connection end was 50 ft. The Timken bit with chrome-plated thread averaged 1200 ft, and rod life increased to as much as 500 ft. However, the life of the connection end was much better on shorter length drill rods or in places where machines with 34-in. change were used.

The bit thread continued to be the point of ultimate failure with thread strippage, constituting the cause for discard of bits. In one of the new development headings, harder rock was encountered for approximately 800 ft, dropping the life per bit to a low of 90 ft with shank and thread life of rods dropping to approximately 125 ft average.

The stripped bits were then welded to the rods, increasing the life per bit by 75 to 100 pct. The rod transportation for main level development was not a problem so intraset rods were tried. Intraset rods have tungsten carbide inserts set into the rods proper by the manufacturer and can be obtained with chisel or four point bits. This type of rod eliminates the need for any connection and the steel being a special alloy will show more feet drilled per rod. The first trial was made with eight rods, and final results averaged 350 ft per rod, six of the rods worked the life of the bit end, and two broke shanks at less than 50 ft.

The preceding example showed a considerable improvement, so additional steel of the same type was purchased, but its use has been limited to main level drifting only, because of the handling problem involved in transportation of the complete rod to mine shops for resharpening.

Further trials are being made on improving the life per detachable bit by chrome plating. To date, the chrome plating shows an improvement of approximately 100 pct. However, final results will not be known until the present long term trials have been completed.

Mather Mine "B" Shaft

In November 1947, tungsten carbide bits were first tried at the Mather mine "B" shaft. The use of 1½-in. Carset 13 Series bits, for drilling the 72-hole, 7-ft shaft round, decreased the drilling time from an average of 4½ hr per round required with steel bits, to 2 hr with insert bits. The best drilling time for

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the tungsten carbide bits on an 8-ft round was 50 min, and it never exceeded 3½ hr. The maximum drilling time for steel bits was 11 hr, using 1700 jackbits for a 4-ft round. The steel bits averaged four uses per bit for a total of 6.2 ft per bit, whereas the tungsten-carbide bits averaged seven uses with total average of 135 ft drilled per bit.

The rods and bits were inspected at the shop before every drilling period.

The bits were gaged, and if found to have lost more than 1/16 in., were removed. The threads were also checked, and if sharpening of the thread contour was evident, the bits were discarded from shaft use. However, in development other than shaft sinking, these bits were used again, thus attaining an additional 50 pct life in feet per bit before final discard.

In rock development where Jumbo-mounted, 3-in. drifters were employed, the 1½-in. 13 Series bit did not prove satisfactory. The attachment end failures duplicated those explained in the example for the Mather mine "A" shaft. The threaded rod with 115 Series Carset special hard insert bit was tried and proved to be 100 pct better in feet per bit and approximately 500 pct better in feet per rod connection end. The entire mine was converted to 115 series with a result of 300 to 500 ft per bit, varying with the nature of the rock. Further experimentation with other types of bits or connections has been discontinued, at least temporarily, in favor of the standardization effected.

Greenwood Mine

Tungsten carbide trials at the Greenwood mine were started early in 1948 using various competitive types and makes of bits in conjunction with jackleg-mounted jackhammers. For drift work, 2⅝-in. jackhammers were used attached to pneumatic reverse feeds mounted on either Jumbos, or posts and arms. They also experienced similar connection end failures. By changing to a slower rotation on the up stroke, the life of the bit was improved by approximately 50 pct. Strippage continued to be the principal problem. Chrome plating of the bit threads was tried, with improvement, but later approximately the same results were obtained by turning the rod thread 0.002 in. oversize.

The results of the early trials are as follows: 1—Type A bit footage, drilled 1948, 55,490 ft; average per bit, 192 ft. 2—Type B bit footage, drilled 1948, 51,521 ft; average per bit, 163 ft. 3—Type A bit footage, drilled 1949, 127,000 ft; average per bit, 291 ft.

The total mine average for the year 1950 is 145 ft per bit with type A and C bits. The lower footage figure is attributed to the additional hard rock work encountered, which is charged for approximately 50 pct of the total bits for the year. The comparative figures are as follows: 1—Jumbo drifting in hard rock and jasper, 56 ft per bit; 2—Raise development, hard hematite, 259 ft per bit; 3—Stope development, hard hematite, 640 ft per bit; 4—Stope development, hard magnetite, 376 ft per bit.

The penetration speed averages 10 in. per min, varying from 6 in. to 15 in.

The increase in feet per bit drilled for the years 1949 and 1950 over that of 1948 is credited to miner training which included: 1—Insistence on the tightening of bits on rods with Stillson wrenches after each use. 2—Instruction that the machine should not be turned on until the bit is tight against bottom of hole when changing to longer lengths. 3—Orders that bits be returned for reconditioning when 3/32 in. wear shows on cutting edge. 4—The use of 0.005 in. undersize ring gages issued to the miner to control rod discard. 5—The use of ring gages by shift bosses to double check the miners' rods at least once a day.

The footage per bit in stopes and development drifts is quite satisfactory. The greatest amount of discard is caused by insert wear. Strippage accounts for approximately 10 pct of the discarded bits.

Raise development, however, in 1949, continued to keep the mine average per bit at a lower figure, because it is very difficult to keep a stoper machine tight to the end of the hole. A special stoper was made in the mine shops, with cylinder bore of the pneumatic leg increased in size from 2⅞ in. to 3⅛ in. The results to date show exceptional improvement, and the larger leg is now standard equipment for raise work in the Greenwood mine. However, with the added push on the leg it is necessary to use greater care in the construction of the drilling stages to compensate for the additional pressure.

Champion Mine

The Champion mine was reopened in April of 1948, and the first tungsten carbide bits tried were the 1¾-in., 15 Series Carset stud connection and 1½-in., 13 Series Carset stud connection. The average life per bit connection end was 150 ft using the miner training and precautions listed previously for the Greenwood mine. Next the 15 Series 1⅝-in., Carset bit was tried. The mine average was raised to 200 ft per bit, and an additional 50 ft was gained by welding the stripped bits to rods, thus increasing the final mine average to 250 ft per bit.

The Champion mine stopes are generally very near to the shaft and nipping is not a problem. Rods were brought to surface each day to be checked for wear and signs of fatigue. Because of this fact the Intraset rod was tried.

Using 4-ft rods with 1⅜-in. bit ends and 8-ft rods with 1 5/16-in. bit ends, results were as follows: Total drilled in year 1950, 202,985 ft; total average per machine shift, 120 ft; total per rod, 319 ft.; total cost per ft drilled exclusive of drillers labor, $0.0532.

The footage per man shift and inches per minute has increased approximately 10 pct through this use of Intraset rods, which is attributed to the smaller diameter holes and the elimination of the threaded bit end.

Down stroke rotation has been tried, but because of the nature of the ground, penetration speed dropped approximately 10 pct, thus making it impractical. One in 40 up stroke is now being used as standard rotation with penetration speed of 12 in. to 18 in. per min.

Cliffs Shaft Mine

Tungsten-carbide bit drilling was started at the Cliffs shaft mine in February 1948, used in conjunction with 3½-in. drifters and jackhammers with jackleg feeds. After a short trial period, it was learned that the available tungsten-carbide bits could not be used economically with the large 3½-in. handcrank, tripod-mounted drifter. Therefore, the experiments for the most part were limited to the use of the bits with 2½-in. and 2⅝-in. cylinder bore jackhammers mounted on a jackleg feed.

In the early tests, each bit was run to destruction without sharpening. Although the average total life during that experiment was 130 ft per bit, the practice was changed because the penetration rate which averages 8 in. per min dropped considerably after the first 60 to 75 ft of drilling, because of excessive dulling of the bits, accompanied with complete insert failure. To remedy this condition, the bits were resharpened after a $\frac{1}{8}$-in. flat appeared on the cutting edge. This was found to be the point to which the bit could be dulled before the insert would crack longitudinally, thereby causing destruction of the insert.

In spite of the care exercised in keeping bits sharp, a great deal of difficulty was experienced in the shattering of inserts during this early period. Finally, with the cooperation of the manufacturers and mine supervisors, this problem was nearly eliminated.

With the improvement of inserts to eliminate shattering, the total footage per bit increased considerably, bringing in a new problem, namely, connector thread strippage. This continued to be the main difficulty until early 1949.

The average insert bit cost per foot drilled for the year 1948 was $0.0898. Although slightly higher than the comparable steel bit costs, the obvious savings effected by small hole drilling made it advantageous to convert to tungsten carbide bits for general use and continue further experiments to improve the technique and bits.

At the beginning of 1949 the problems encountered with insert bit drilling were as follows: 1—Insert breakage approximately 25 pct caused by: (A) Longitudinal cracks apparently inherent in the available tungsten carbide inserts. (B) Complete breakage caused by use of too large machines. (C) Complete or partial destruction due to miners' own neglect. 2—Thread strippage 75 pct caused by: (A) Complete strippage where excessive tolerances were allowed by manufacturers on bit threads and rod stud threads. (B) Wear caused by tight and loose running of bit.

The efforts to eliminate thread strippage were as follows: 1—Thread contour gages were made to show the miner when he should discard his rod. This procedure resulted in culling 75 pct of the rods in use at that time. 2—Partly stripped bits were culled with hopes of preventing rod thread wear. The worn bits would, of course, wear a new rod and as a result, that worn rod helped wear or strip a new bit. 3—The shift bosses were given "no go" ring gages with which to check the rods at least once a day.

These three precautions resulted in improvement of bit appearance, footage drilled per man shift, and feet per bit; however, strippage continued to be the point of ultimate failure.

By pure circumstance, a down stroke rotation machine was placed in one of the contract headings. Exceptional bit life and no thread failure resulted. Because of this discovery, more machines were changed to down stroke rotation in those contract faces where excessive stripping of bits and rods was encountered. Bit life increased 50 to 100 pct with 100 pct elimination of thread strippage. Down stroke rotation is being used exclusively at the Cliffs shaft mine at the present time. However, a new problem was introduced, that of thread stud breakage. Often a bit must be discarded prematurely because the broken stud cannot be removed.

The change to minimize this difficulty was 1—slow down the down stroke rotation to a ratio of 1 in 40 which caused the point of ultimate failure to occur in the rod section, rather than the stud thread, and 2—discontinue the use of the stud 13 and 15 Series connection and convert to use of field-threaded rod for the 4-point, 115 Series $1\frac{5}{8}$-in. Carset bit or the 113 Series $1\frac{3}{8}$-in. Carset bit, which have proved most satisfactory for Cliffs shaft mine use.

Table I. Drilling Results

Year	Total Ft Drilled	Avg Ft Per Bit	Cost Per Ft Drilled
1948	125,218	125	.092
1949	281,148	170	.0713
1950	627,276	175	.077
January through March	149,748	188	0.0731
April through June	163,494	160	0.0841
July througn September	147,250	112	0.1127
October through December	166,784	385	0.0405

During the first 9 months of 1950, drilling results steadily dropped in feet per bit, thus raising the cost per foot drilled. Table I gives the drilling results for 1948 through 1950. The 3-month periods for 1950 show an actual drop in footage per bit until the end of August, which is attributed to stud breakage in skirt of bit causing premature discard of the bits. The rise in feet per bit for the last 3 months is attributed to the discontinuance of the 13 Series stud and change over to the heretofore mentioned 115 Series threaded rods with $1\frac{5}{8}$-in., 115 Series Carset bit.

In addition, the sharpening technique was changed. In place of the bench grinder, a J-3 Ingersoll-Rand Jackbit grinder was converted to insert bit sharpening. The use of the grinder provided more accurate control of gage and cutting angle. The wheel used is a Simons Worden White Co. 12x1x$1\frac{1}{4}$-in. type, IK Grain, GC60-J14V14, or the Carborundum Co.'s silicon carbide wheel GC60-K11-VR and GC60-J8-VW. Other types of wheels have been tried, but the listed ones have proved most practical and satisfactory.

Summary

It is agreed by mine operators that future tungsten carbide drilling will of necessity be classed in two separate groups; the first being hard rock and hard ore drilling with speed of penetration from 3 in. to 15 in. per min and the second group, although limited at present on the Marquette range to 3-in. drifters, will use $3\frac{1}{2}$-in. cylinder bore drifters for development in the future where speed is most important.

To gain greater penetration speed in hard materials of the first group, experiments with smaller diameter holes are now in progress using both detachable bits and rods with tungsten carbide inserts set into the rod proper. The indicated results are very favorable although final results will not be known for some time.

Experiments are also being made on the Marquette range with rotary drilling using tungsten carbide tipped auger steel. The results of the tests prove more satisfactory than the percussion type auger drilling now being used, and it is possible that the rotary drilling of soft hematite will be used much more extensively within the next year.

The various machine changes and technique precautions herein mentioned have improved, to a great extent, the footage per bit. However, it is the combined opinion of all users that better results will be realized as experiments are continued.

Acknowledgment

Gratitude is expressed to the Management and Personnel for their cooperation in making this paper possible.

Five Variable Flotation Tests Using Factorial Design

by Adrian C. Dorenfeld

Factorial design is a mathematical method of drawing valid conclusions from a series of tests made in a predetermined pattern. It is applied to flotation ore testing using, in this case, five variables, each varied twice, requiring a total of 32 tests. Representative conclusions are checked by repeat testing.

THE factorial design method of testing is particularly applicable to mill-scale plant tests because 1—the conclusions take into consideration all variabilities of the plant, 2—there are no subjective guesses as to the conclusions, 3—the number of tests necessary to check large numbers of variables are at a minimum, and 4—interactions between variables can be discerned. It is equally applicable to laboratory testing because once the variables to be investigated have been decided upon, as well as the number of levels for each variable, a well-trained assistant can run the tests, since the test work follows a predetermined mathematical pattern, leaving the conclusions to be found by the person in charge. The conclusion takes into consideration all errors of the tests. To illustrate the use of factorial design, a sample of copper porphyry ore was tested, the copper content of the batches not being uniform. Five variables were investigated, each at two levels, requiring only 32 tests. After suitable arithmetical analysis, conclusions for copper recovery and concentrate grade are arrived at. Four cases of these conclusions are checked by repeat testing.

Testing Methods

Reproducibility of flotation tests varies widely, depending on the experimenter, the process, the equipment, and probably a host of unknown and perhaps interacting variables. If the reproducibility of tests is good, let us say to 0.2 pct recovery of a certain metal, and the minimum difference between tests upon the variation of the interested variables is, say 1 pct, then simple averaging and inspection is ample in interpreting results.

However, often such a happy situation is not the case. This is particularly true in mill-scale operations where the ores are variable, and a host of minor operating fluctuations common to most milling, and familiar to all mill operators, is the normal state of affairs. Yet, mill scale tests are run, the results averaged, and conclusions drawn therefrom. The trustworthiness of such results should be measured by comparison to the experimental error—the measure of the degree of reproducibility of the results. This comparison is a test of significance. Thus, if daily results of recoveries average 80.0 pct, and a spread of 78.0 pct to 82.0 pct, is recorded, then, clearly, on changing mill operations, an 81 pct recovery cannot be construed as significant by using averages, since it is well within the previous test limits. By using rigorous statistical tests of significance, conclusions arrived at by inspection or simple averaging may be shown to be based on inadequate evidence or even wrong. In any event, a plant super-

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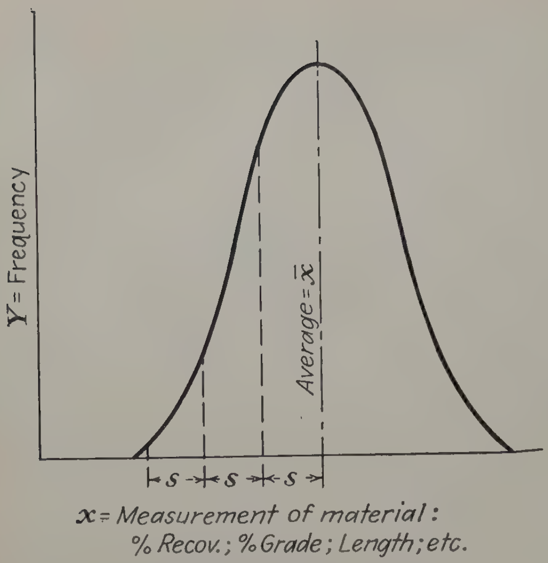

Fig. 1—Normal distribution curve.

intendent would like to know whether the recommendations, based on tests, can be shown to be scientifically sound, or are merely subjective guesses.

There are methods of testing statistically whether one method of operation is different from another. They are based on the use of relatively large numbers of tests and the variation of only one variable at a time. Therefore, to test as many as five variables a large number of tests would be required. The number of tests would be dependent on the reproducibility of results. However, there is a method, long used in agricultural experimentation and adopted for some years in the chemical industries, that uses a limited number of tests, executed in a mathematical pattern, and yields the following information: the significance of changes in relation to experimental error; the probable average and range of results; and whether there are interactions between variables or not. This method, called factorial design, makes use of the results of one test several times.

Definition of Terms

1—Normal distribution is a mathematical distribution made up of individuals perfectly described by knowing the average and standard deviation of these individuals, see Fig. 1.

2—Mean or average is the line which divides a distribution area in exactly equal parts, see Fig. 1. It is the moment of the data about the origin.[1]

3—Variance is a measure of variation of the data about the mean. It is the second moment of the data about the mean.[2]

4—Standard deviation is the square root of the variance.[3] It is given by the formula:

$$\text{standard deviation} = S = \sqrt{\frac{\Sigma X^2}{n} - \overline{X}^2} \qquad [1]$$

where: X^2 = the sum of the square of each individual term of the data
n = number of individuals in the data
$\overline{X}$ = the average of the data

5—Population or universe is the infinite number of tests that could be made using one particular physical condition. The greater the number of tests the closer the approach is to the population. The mean of the population is denoted by m, and the standard deviation by σ.

6—Sample is a finite number of tests, N, drawn at random from a population. Each sample will have its standard deviation, S, and average $\overline{X}$.

7—Significance level is the risk taken in being wrong. Thus a significance level of 5 pct means that in the long run the conclusion may be wrong five times out of 100 times. It is usually fixed by the economics of the situation. If large-scale costly additions would be necessary to install a new process, then a significance level of 1 pct or better would be justified; if the new process is merely a change in reagents, of substantially equal costs, then a significance level of perhaps 20 pct might be justified.

8—F distribution is the distribution of variances. Tables for this distribution exist.[4] If the following ratio:

$$F = \frac{NxSx^2/Nx\text{-}1}{NySy^2/Ny\text{-}1}$$

$$\nu 1 = Nx\text{-}1$$

$$\nu 2 = Ny\text{-}1$$

where
Nx = number of individuals in sample X

Table I. Tests on Copper Porphyry Ore

Factors	Symbol	Level 1	Level 2
Grind, mesh	G	−65	−200
pH	P	8.5	10.5
Xanthate, lb per ton	X	0.20	0.30
Sodium cyanide, lb per ton	C	0.0	0.10
Frother (cresylic acid), lb per ton	F	0.05	0.10

Table II. Tests on Copper Porphyry Ore

Test No.	G Mesh	P	X, Lb per Ton	C, Lb per Ton	F, Lb per Ton	Cu Recovery, Pct	Cu Concentrate Grade, Pct	Cu in Heads, Pct
1	65	8.5	0.2	0.0	0.05	21.21	5.65	0.92
2	65	10.5	0.2	0.0	0.05	70.58	17.40	0.95
3	65	8.5	0.3	0.0	0.05	49.33	6.19	0.88
4	65	10.5	0.3	0.0	0.05	60.65	12.47	0.83
5	200	8.5	0.2	0.0	0.05	17.28	1.95	0.86
6	200	10.5	0.2	0.0	0.05	85.43	14.75	1.03
7	200	8.5	0.3	0.0	0.05	21.82	3.67	0.82
8	200	10.5	0.3	0.0	0.05	75.30	8.73	0.89
9	65	8.5	0.2	0.10	0.05	27.26	6.90	0.93
10	65	10.5	0.2	0.10	0.05	64.78	19.62	0.99
11	65	8.5	0.3	0.10	0.05	60.87	11.00	0.88
12	65	10.5	0.3	0.10	0.05	66.67	13.58	0.78
13	200	8.5	0.2	0.10	0.05	25.14	5.82	0.84
14	200	10.5	0.2	0.10	0.05	75.69	10.13	0.87
15	200	8.5	0.3	0.10	0.05	39.83	8.52	0.92
16	200	10.5	0.3	0.10	0.05	69.89	10.67	0.91
17	65	8.5	0.2	0.0	0.10	57.65	6.58	0.95
18	65	10.5	0.2	0.0	0.10	76.35	11.86	0.92
19	65	8.5	0.3	0.0	0.10	47.64	4.53	0.85
20	65	10.5	0.3	0.0	0.10	74.40	8.14	0.70
21	200	8.5	0.2	0.0	0.10	39.99	2.35	0.70
22	200	10.5	0.2	0.0	0.10	77.97	4.72	0.58
23	200	8.5	0.3	0.0	0.10	49.64	3.51	0.75
24	200	10.5	0.3	0.0	0.10	75.93	6.71	0.73
25	65	8.5	0.2	0.10	0.10	62.87	18.91	0.99
26	65	10.5	0.2	0.10	0.10	74.84	17.40	0.92
27	65	8.5	0.3	0.10	0.10	60.61	17.32	0.91
28	65	10.5	0.3	0.10	0.10	72.07	13.90	0.79
29	200	8.5	0.2	0.10	0.10	70.66	8.14	0.76
30	200	10.5	0.2	0.10	0.10	82.61	8.81	0.80
31	200	8.5	0.3	0.10	0.10	63.29	10.40	0.73
32	200	10.5	0.3	0.10	0.10	86.12	12.66	0.98

Avg = 0.855
Standard Deviation, pct = 0.10

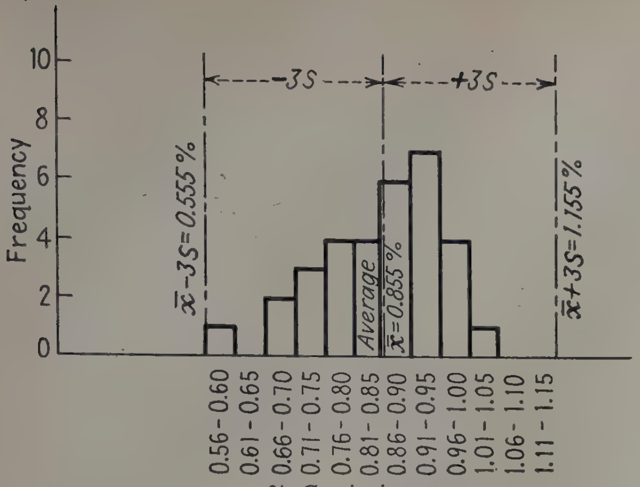

Fig. 2—Grade of heads, pct Cu, 32 tests.

Ny = number of individuals in sample Y
S^2x = variance of sample X
S^2y = variance of sample Y
$\nu 1$ = degree of freedom of X
$\nu 2$ = degree of freedom of Y

is smaller than the ratio in the F distribution table for the particular level of significance, then X and Y are samples drawn from the same population, and in the long run will be equal a greater percentage of the time than the level or significance used.[5]

Experimental

A sample of a copper porphyry ore was obtained and crushed to —10 mesh. The ore was not thoroughly mixed because a variable ore was wanted to simulate conditions in practical milling. One thousand gram charges of the ore were made up. The charges for testing were selected at random. Five variables were investigated at two levels. The results are given in Table I.

The mathematical pattern calls for 32 tests being run. The results of these tests are given in Table II.

Factorial design of an experiment is the planning of tests in such a way that the result of each test can be used several times. It is based on the premise that the results of the experiment are normally distributed, and therefore the equations for this distribution apply. From the Tchebycheff inequality, it has been shown that for total nonnormality, the greatest error possible at 2σ limits is 25 pct, if the normal distribution equations are applied to nonnormal distributions.[6] Further, Camp[7] and Meidell[8] have shown that if a distribution is such that it falls off rapidly from the mean, the greatest error possible is 11.1 pct, at 2σ limits, in applying the normal distribution equations. For normal distributions, the greatest error possible is 4.5 pct.[6]

Fig. 2 shows a graph of the distribution of copper content of the heads and of one series of concentrates. They fulfill the conditions of Camp-Meidell, and therefore the normal distribution equations can be used with confidence. Since the balance of the tests are similar in nature, then by analogy, their distributions will be similar. Hence, the method of factorial design can be safely applied.

For normal distributions, variances are addative.[9] Thus, if a series of tests is run varying only one variable, then the variance is S_1^2. Similarly the variance of variable S_2 is found, etc. The variances of all the tests is the sum of all the single variable variances S_1^2, S_2^2--S_n^2 or:

$$S^2 \text{ Total} = S_1^2 + S_2^2 + S_3^2 + \text{--}S_n^2 \qquad [2]$$

Another method for obtaining the total variance is to consider all the tests as one series and obtain it in the usual way as described in the definition of terms. If the tests are perfect, the variances will be equal. This is seldom the case. The difference between the two total variances is the variance of all unknown factors—it is the variance of the experimental error.[10, 11] From this fundamental concept, the method for testing component variances is built up. Further mathematical proofs are in the texts listed in the references.[1, 9] The mechanics of the analysis for the recovery of copper follows.

The mathematical pattern for testing is merely that all possible combinations of tests are run. In this case, for a five variable test at two levels each, it is 32 tests, see Table III.

Eq 1 showed that the standard deviation is:

$$S = \sqrt{\frac{\Sigma X^2}{n} - \overline{X}^2}$$

$$\text{or } S^2 = \frac{\Sigma X^2}{n} - \overline{X}^2$$

since the average $\overline{X}$, is the sum of all the individuals, divided by the number of such individuals, or $\overline{X} = \frac{\Sigma X}{n}$, we can write:

$$S^2 = \frac{X^2}{n} - \frac{(\Sigma X)^2}{(n)^2} \qquad [3]$$

multiplying by n, we get:

$$nS^2 = \Sigma X^2 - \frac{(\Sigma X)^2}{n} \qquad [4]$$

From the definition of terms, the F ratio $(NxSx^2/n\text{-}1) \div (NySy^2/n\text{-}1)$ is a test of significance whether sample x and y are randomly related or not, it is clear interest should be centered on the nS^2 terms. This is called the sum of squares. The $nS^2/n\text{-}1$ terms are the unbiased variances.[12] To obtain the sum of squares, eq 4 shows that it is necessary to square

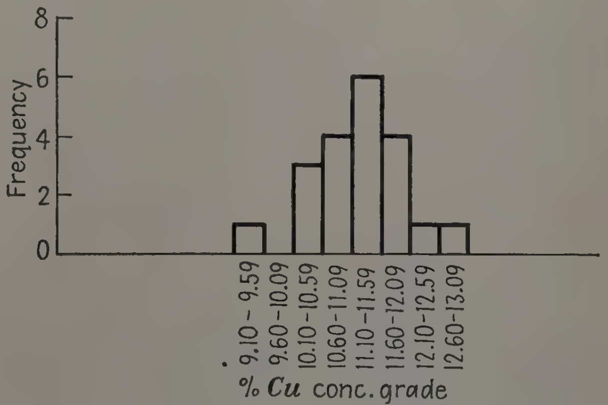

Fig. 3—Grade of concentrate, pct Cu, 20 repeat tests of test 32.

Table III. Copper Recoveries of Tests

		G_1 = 65 Mesh Grind				G_2 = 200 Mesh Grind			
		X_1 = 0.2 Lb per Ton Xanthate		X_2 = 0.3 Lb per Ton Xanthate		X_1 = 0.2 Lb per Ton Xanthate		X_2 = 0.3 Lb per Ton Xanthate	
		P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5
F_1 = 0.05 lb per ton	C_1 = 0.0 lb per ton NaCN	21.21	70.58	49.33	60.65	17.28	85.43	21.82	75.30
Frother	C_2 = 0.1 lb per ton NaCN	27.26	64.78	60.87	66.67	25.14	75.69	39.83	69.89
F_2 = 0.10 lb per ton	C_1 = 0.0 lb per ton NaCN	57.65	76.35	47.64	74.40	39.99	77.97	49.64	75.93
Frother	C_2 = 0.1 lb per ton NaCN	62.87	74.84	60.61	72.07	70.66	82.61	63.29	86.12

Table IV
$G_1 = G_2$

		X_1 = 0.2 Lb per Ton Xanthate		X_2 = 0.3 Lb per Ton Xanthate	
		P_1 = pH8.5	P_2 = pH10.5	P_1 = pH8.5	P_2 = pH10.5
F_1 = 0.05 lb per ton	C_1 = 0.0 lb per ton NaCN	38.49	156.01	71.15	135.95
Frother	C_2 = 0.1 lb per ton NaCN	52.40	140.47	100.70	136.56
F_2 = 0.10 lb per ton	C_1 = 0.0 lb per ton NaCN	97.64	154.32	97.28	150.33
Frother	C_2 = 0.1 lb per ton NaCN	133.53	157.45	123.90	158.19

each individual and add; subtract from this the square of the sum divided by total number of tests. This last, $\frac{(\Sigma X)^2}{n}$, is called the correction factor. It is actually part of the arithmetic in arriving at the variance. Similar calculations are carried out in arriving at the component variances. The mathematics used to separate the component variances from the total number of tests are described by P. G. Hoell.[13, 14]

The mechanics of using the test results of Table III follows: The sum of the tests are squared and divided by the number of tests:

$$\frac{(\Sigma X)^2}{n} = \frac{1{,}904.37^2}{32} = 113{,}332.03$$

This is the correction factor, (C.F.) already discussed. Each individual test result is squared and summed: $\Sigma X^2 = 125{,}734.63$. The total sum of squares is $\Sigma X^2 - \frac{(\Sigma X)^2}{n}$ or 12,402.60. Each variable is summed over in turn. This means that the assumption is made that in the case of grinding, the difference in results between 200 mesh grinding and 65 mesh grinding is no more than would be made if duplicate tests at each grind were made. Should this not be the case, then in the final analysis of variance, the variances of each component will be significantly greater than the experimental error variance. Yet, this preliminary assumption does not invalidate the final results.[15] Evidence obtained by repeat testing will be presented that this is physically the case. As an example, assume the results of 65 mesh grinding, G_1, are but an experimental variant of 200 mesh grinding, G_2, or $G_1, = G_2$. Then test $F_1\,C_1\,P_1\,Q_1\,G_1$ is an experimental variant of test $F_1\,C_1\,P_1\,Q_1\,G_2$ and can be added together. Thus Table IV is built up.

Similarly four other tables are built up, letting $X_1 = X_2$; $P_1 = P_2$; $C_1 = C_2$; $F_1 = F_2$.

The next step is to sum over the variables two at a time. This is done by taking each table as built up in Table IV, and assuming each variable to be equal, or as in Table IV, $X_1 = X_2$. Such a table is illustrated in Table V.

Similarly nine other such tables are computed. The next step is summing over the variables three at a time, or in this illustration, allowing $G_1 = G_2$; $X_1 = X_2$; $P_1 = P_2$. Table VI is thus computed:

Nine such tables can then be computed. The sum of squares of each component can now be obtained. The correction factor is known. The mechanics follow:

Table V
$G_1 = G_2$; $X_1 = X_2$

P_1 = pH8.5				P_2 = pH10.5			
C_1 = 0.0 Lb per Ton NaCN		C_2 = 0.1 Lb per Ton NaCN		C_1 = 0.0 Lb per Ton NaCN		C_2 = 0.1 Lb per Ton NaCN	
F_1 = 0.05 lb per ton Frother	F_2 = 0.10 lb per ton	F_1 = 0.05 lb per ton	F_2 = 0.1 lb per ton	F_1	F_2	F_1	F_2
109.64	194.92	153.10	257.43	291.96	304.65	277.03	315.64

Table VI

$G_1 = G_2;\ X_1 = X_2;\ P_1 = P_2$

	$C_1 = 0.0$ Lb per Ton NaCN	$C_2 = 0.10$ Lb per Ton NaCN	Totals
$F_1 = 0.05$ lb per ton Frother	401.60	430.13	831.73
$F_2 = 0.10$ lb per ton Frother	499.57	573.07	1072.64
Totals	901.17	1003.20	1904.37 Grand Total

Table VII. Variances for Cu Recoveries

Type Reaction	Source of Variance		Degrees of Freedom $(n\text{-}1)$	Sum of Squares (nS^2)	Variance $\frac{(nS^2)}{(n\text{-}1)}$
Main Effect	Grind =	G	2−1 = 1	2.43	2.43
	Xanthate =	X	2−1 = 1	59.82	59.82
	pH =	P	2−1 = 1	7,026.76	7,026.76
	NaCN =	C	2−1 = 1	325.33	325.33
	Frother =	F	2−1 = 1	1,813.68	1,813.68
First Order Reaction	G x X		(2−1) (2−1) = 1	27.47	27.47
	G x P		(2−1) (2−1) = 1	515.12	515.12
	G x C		(2−1) (2−1) = 1	44.43	44.43
	G x F		(2−1) (2−1) = 1	29.54	29.54
	X x P		(2−1) (2−1) = 1	301.28	301.28
	X x C		(2−1) (2−1) = 1	23.20	23.20
	X x F		(2−1) (2−1) = 1	154.13	154.13
	P x C		(2−1) (2−1) = 1	377.50	377.50
	P x F		(2−1) (2−1) = 1	597.80	597.80
	C x F		(2−1) (2−1) = 1	63.19	63.19
All other Residual Interactions 10 third order 5 fourth order 1 residual			(10+5) + (2−1) (2−1) (2−1) (2−1)) (2−1) = 10+5+1 = 16	1,040.93	65.06
Total				12,402.60	

The sum of squares due to cyanide is:

$$\frac{\overline{901.17}^2 + \overline{1{,}003.20}^2}{16} = 113{,}657.35$$

$$\text{Correction Factor} = \underline{113{,}332.03}$$

$$\text{Sum of squares due to cyanide} = 325.32$$

Similarly the sum of squares due to each factor is obtained.

There is always a possibility that the reaction between two variables is the factor responsible for the results. Physically, this is the case for cyanide and hydroxyl ion concentrations. It has been amply illustrated that the cyanide ion concentration is the governing factor in copper depression. This ion concentration can be varied by varying the pH. Yet the pH alone, within limits, has no bearing on copper depression. To obtain such interactions, each term of Table VI is squared and divided by the number of terms composing it. As an illustration, the sum of squares for the interaction between cyanide and frother would be:

$$\frac{\overline{401.60}^2 + \overline{430.13}^2 + \overline{499.57}^2 + \overline{573.07}^2}{8} \text{ or } 115{,}534.22.$$

Sum of squares due to cyanide—	325.32
Sum of squares due to frother—	1,813.68
Correction Factor	113,332.03
Sum of squares due to interaction of cyanide and frother	63.19

Similarly the other interactions are obtained. Second order interactions are also possible. These would be reactions among any three of the variables. There might also be third order interactions as well. However, in flotation testing, the possibility of interactions greater than first order is probably small, from a consideration of the chemistry involved. However, in poorly known processes, all interactions should be investigated. A complete set of variances of all interactions will be presented in an analysis of concentrate grade.

The variances shown in Table VII then were obtained.

Residual sum of squares = total sum of squares — main effect squares — first order squares.

All the first order interaction variances numerically less than the residual — G x X, G x C, G x F, X x C, C x F, are not significant. The experimental error is greater than these variations. Therefore, the variations of these interactions are a part of the experimental error.[16] The results of using the F dis-

Table VIII. F Test for First Order Interactions

Interaction	Variance	Interaction ÷ Residual	5 Pct Level of Significance from Tables
X x F	154.13	2.37	4.49
X x P	301.28	4.44	4.49
P x C	377.50	5.80	4.49
G x P	515.12	7.92	4.49
P x F	597.80	9.19	4.49

Table IX. Variances at P_1 and P_2 Levels
Two 4 Factor Experiments

Type Reaction	Source of Variance	Degrees of Freedom $(n\text{-}1)$	Sum of Squares (nS^2)	Variance $\frac{(nS^2)}{(n\text{-}1)}$
At P_1 Level (pH = 8.5).				
Main Effect	G = Grind	(2−1) = 1	223.42	223.42
	X = Xanthate	(2−1) = 1	314.79	314.79
	C = NaCN	(2−1) = 1	701.85	701.85
	F = Frother	(2−1) = 1	2246.99	2246.99
First Order Reactions	G x X	(2−1) (2−1) = 1	48.84	48.84
	G x C	(2−1) (2−1) = 1	74.01	74.01
	C x F	(2−1) (2−1) = 1	152.60	152.60
	X x C	(2−1) (2−1) = 1	2.54	2.54
	X x F	(2−1) (2−1) = 1	517.00	517.00
	C x F	(2−1) (2−1) = 1	22.69	22.69
Four Second Order Interactions + Residual	Residual	5	341.49	68.29
At P_2 Level (pH = 10.5)				
Main Effect	G	(2−1) = 1	294.12	294.12
	X	(2−1) = 1	46.31	46.31
	C	(2−1) = 1	0.97	0.97
	F	(2−1) = 1	164.48	164.48
First Order Interactions	G x X	(2−1) (2−1) = 1	50.18	50.18
	G x C	(2−1) (2−1) = 1	0.66	0.66
	G x F	(2−1) (2−1) = 1	21.77	21.77
	X x C	(2−1) (2−1) = 1	27.25	27.25
	X x F	(2−1) (2−1) = 1	29.83	29.83
	C x F	(2−1) (2−1) = 1	41.99	41.99
Four Second Order Interactions + Residual	Residual	5	52.06	10.41

Table X. F Test for First Order Interactions, P_1 Level

Interactions	Variance	Interaction Variance Residual ÷ Variances	5 Pct Level Value from Tables
G x C	74.01	1.1	6.61
C x F	152.60	2.2	6.61
X x F	517.00	7.6	6.61

Table XI. First Order Reactions Grouped with Residual

Source of Variance	Degree of Freedom	Variance
G	1	223.42
X	1	314.79
C	1	701.85
F	1	2246.99
X x F	1	517.00
Residual	10	64.22

tribution to test whether the variances greater than the residual are significant, are given in Table VIII.

From Table VIII, the only interaction not significant is that of X x F—the other four reactions are significant. These four reactions have one common factor, pH, or P. This may mean that the interactions are significant merely because the main effect, P, is very large. To test this, P can be eliminated by breaking down the five factor experiment to two four factor experiments; one series of tests at P_1 level, (pH8.5) and the other series at P_2 level, (pH10.5). Table IX is then obtained.

Consider the results at the P_1 level. Only the following interactions are numerically greater than the residual, see Table X.

Only the X x F interaction is significant. The results of grouping the nonsignificant first order interactions with the residual are shown in Table XI.

The G and C main effects, to exist, must be significantly greater than the residual; the X and F effects must each be significantly greater than the X x F interaction, because the X and F main effects are each made up of the X x F variance plus the main effect variances.[17] Applying the F test for the above table, Table XII is obtained.

Therefore, the conclusion is that the X, F, and G effects do not exist. Variation of grind, xanthate or frother do not, by themselves, affect the results. Differences caused by changes in these physical variables are merely experimental errors. However, xanthate and frother changes affect each other; using 0.2 lb per ton xanthate and 0.05 lb per ton frother cannot be compared to 0.2 lb per ton xanthate and 0.10 lb per ton frother on the basis that superior results of the latter are caused by the increase of frother—that frother acting only as a frother has increased the recovery of the metal. The tests show that at a pH of 8.5, the frother reacts with the xanthate, affecting the collection of copper. It is the reaction between the two that causes the results. Physically, this is explainable by previously published results and will be discussed in detail after the conclusions are given.

Table XII. F Test Applied for Table XI

Source of Variance	Source of Variance ÷ Residual Variance or the Interaction Variance	5 Pct Level from Tables
G	3.5	4.96
C	10.9	4.96
X	Numerically $<X$ x F	161.0
F	4.4	161.0

The only significant main effect is cyanide. The other effect is the xanthate-frother interaction. All tests that have the three variables in common: cyanide, xanthate, and frother, at any one level, can be pooled, because the other variable, grinding, has been shown to be nonsignificant. It does not affect the results more than by chance causes—the experimental error. It is immaterial whether grinding to 65 mesh or to 200 mesh is practiced. Any difference due to this change is within the error of results caused by ore variability, process variability, sampling variability, assay variability, etc. The conclusions are summarized in Table XIII.

Table XIII. Conclusions at pH8.5 Variability of Grinding Nonsignificant

Conclusion	Significant Variables, Lb per Ton			Test No. in Table II	Recovery Average Cu, Pct	Standard Deviation Cu, Pct
	Xanthate	Frother	Cyanide			
1	0.2	0.05	0.00	1,5	19.25	2.04
2	0.2	0.05	0.10	9,13	26.45	1.31
3	0.2	0.10	0.00	17,21	48.82	8.83
4	0.2	0.10	0.10	25,29	66.77	3.81
5	0.3	0.05	0.00	3,7	35.58	13.75
6	0.3	0.05	0.10	11,15	50.35	10.54
7	0.3	0.10	0.00	19,23	48.64	1.00
8	0.3	0.10	0.10	27,31	61.95	1.34

It will be noted that some of the conclusions yielded results with large standard deviations. These results did not have good reproducibility, particularly conclusions 5 and 6. These tests used 0.3 lb per ton xanthate and 0.05 lb per ton frother. The mathematics show an interaction occurred. Physically, these froths were scummy and exceptionally thin. Pulp showed throughout these froths. The reaction can be explained by the fact that a substantial part of the frother, cresylic acid, could have been attached to the sulphide surfaces, the xanthate serving as anchors, and therefore not leaving enough frother for frothing.[18] In conclusions 1 to 4, a smaller amount of xanthate was used, and therefore not as much cresylic acid could have been attached to the sulphide particles to leave enough cresylic acid for frothing purposes. Similarly, in conclusions 7 and 8, 0.1 lb per ton cresylic acid was used, or twice as much as in tests making up conclusions 5 and 6, leaving enough cresylic acid for good frothing. These conclusions yielded good reproducible results because of the ample frothing, which allowed similar amounts of froth to be raked off in any two tests. Conclusion 3 yielded a large standard deviation, not explainable by the above. It could be caused by an error in measurement of reagents, sampling, etc. It does not materially affect, however, the final conclusions.

Analysis of results at pH 10.5, Table IX, shows that none of the interactions are significant. Table IX then becomes Table XIV. The F test of Table XIV given in Table XV.

Table XIV. Main Effects at pH10.5

Main Effect	Degrees of Freedom $(n-1)$	Sum of Squares nS^2	Variances $\frac{nS^2}{n-1}$
G	1	294.12	294.12
X	1	46.31	46.31
C	1	0.97	0.97
F	1	164.48	164.48
Residual	11	223.74	20.34

Table XV. F Test of Table XIV

Main Effect	Main Effect Variance ÷ Residual Variance	5 Pct Level of Significance from Tables
G	14.5	4.8
X	2.3	4.8
C	Less than Residual	4.8
F	8.1	4.8

At a pH of 10.5, xanthate and cyanide variability are not significant. Only variation of grinding and frother affects the results. The conclusions are given in Table XVI.

These conclusions are physically reasonable. At a high pH, the chalcocite-pyrite middlings are depressed. Therefore, fine grinding is necessary to liberate the chalcocite from the depressed pyrite-chalcocite middlings. At this pH, cyanide has little effect on pyrite depression, since the high pH does an efficient job. The lesser amount of xanthate is sufficient to coat the copper minerals, and since the pyrite chalcocite middlings are well depressed, additional xanthate does not affect copper recovery materially. The recoveries were low with the smaller amounts of frother because cresylic acid at the high pH of 10.5 is fairly soluble,[19] and hence not much is available to cause frothing. Doubling the amount of frother increased the frothing and therefore the recovery of copper.

Validity of Conclusions

Conclusion 4 of Table XVI is made up of four tests, two variables of which were shown to be nonsignificant. These variables were cyanide and xanthate. If it could be shown experimentally that the results would have been substantially the same even if these variables were not varied, then the mathematical analysis would be confirmed. Accordingly four tests were run, using the conditions of test 32, Table I. The copper recoveries were: 84.65, 76.02, 78.01, and 83.39, or an average of 80.52 pct and standard deviation of 3.54 pct. This compares with conclusion 4 of Table XVI of 80.66 pct and a 3.92 pct standard deviation. Using the physical conditions of test 22, which conclusion 4 of Table XVI listed as similar to test 32, four repeat tests yielded the following recoveries: 77.02, 82.41, 77.95, 85.67, averaging 81.23 ± 3.38 pct standard deviation. Thus, test 22 is substantially the same as test 32. This was shown to be so by eight tests; factorial design arrived at the same conclusion, by rigorous mathematics, in two tests.

Table XVI. Conclusions at pH10.5
Xanthate and Cyanide Variability Nonsignificant

Conclusion	Significant Variables		Test No. in Table II	Average	Standard Deviation
	Grind, Mesh	Frother, Lb per Ton			
1	65	0.05	2,4,10,12	65.67	2.30
2	65	0.10	18,20,26,28	74.42	1.27
3	200	0.05	6,8,14,16	76.58	5.48
4	200	0.10	22,24,30,32	80.66	3.92

Table XVII. Grade of Concentrate Variances

Type Action	Source of Variance	Degrees of Freedom $(n-1)$	Sums of Squares nS^2	Variance $\frac{(nS^2)}{n-1}$
Main Effect	G = Grind	1	152.73	152.73
	X = Xanthate	1	2.52	2.52
	P = pH	1	153.60	153.60
	C = Sodium Cyanide	1	173.77	173.77
	F = Frother	1	0.03	0.03
First Order Interactions	G x X	1	20.15	20.15
	G x P	1	0.63	0.63
	G x C	1	9.08	9.08
	G x F	1	5.10	5.10
	X x P	1	22.24	22.24
	X x C	1	5.81	5.81
	X x F	1	1.06	1.06
	P x C	1	29.25	29.25
	P x F	1	63.83	63.83
	C x F	1	59.71	59.71
Second Order Interactions	G x X x P	1	4.27	4.27
	G x X x C	1	1.70	1.70
	G x X x F	1	6.88	6.88
	P x C x F	1	0.16	0.16
	P x C x X	1	0.05	0.05
	P x C x G	1	0.19	0.19
	P x X x F	1	18.51	18.51
	C x X x F	1	1.25	1.25
	C x F x G	1	3.35	3.35
	G x C x F	1	5.73	5.73
Third Order Interactions	G x P x C x F	1	12.32	12.32
	X x P x C x F	1	0.02	0.02
	G x X x P x F	1	0.25	0.25
	G x X x P x C	1	0.02	0.02
			3.97	3.97
Unknown Reactions	Residual	1	2.66	2.66
		Total	760.84	760.84

Analysis of Concentrate Grade

The test results of Table II are arranged similar to Table III, but the concentrate grade results are used. The various effects and interactions are obtained in a similar manner. In this case, the analysis was carried out for second and third order interactions as well. The results are shown in Table XVII.

Table XVIII. First Order F Test

Interaction	Variance	Interaction ÷ Residual	5 Pct Significant Level from Tables
G x X	20.15	5.3	4.49
G x P	0.63	Less than residual	4.49
G x C	9.08	2.4	4.49
G x F	5.10	1.3	4.49
X x P	22.24	5.8	4.49
X x C	5.81	1.5	4.49
X x F	1.06	Less than residual	4.49
P x C	29.25	7.7	4.49
P x F	63.83	16.6	4.49
C x F	59.71	15.6	4.49

Table XIX. Four Factor Experiment

Source of Variance	Variance P_1 (pH8.5)	Variance P_2 (pH10.5)
G	66.91	86.45
X	4.88	19.87
C	172.79	30.23
F	30.36	33.50
G x X	2.92	21.50
G x C	5.98	3.29
G x F	10.83	0.00
X x C	2.33	3.54
X x F	5.39	14.19
C x F	33.12	26.75
Residual (5 deg freedom)	3.80	2.68

None of the third order interactions are significant. The new residual is 3.21 with six deg freedom. When measured against this new residual, no second order reaction is significant. They are therefore pooled with the residual, and a new residual estimated: 3.83 with 16 deg of freedom.

Table XVIII shows that G x X, X x P, P x C, P x F and C x F are significant interactions. The frother is involved in two interactions; xanthate in two; pH in two; cyanide in two. This five factor experiment must then be broken down into two four factor experiments, as explained previously. Table XIX gives the results of a test broken down at P_1 (pH 8.5) and P_2 levels (pH 10.5).

Table XX. Significant First Order Interactions

Source of Variance	Variance P_1 = pH8.5	Variance P_2 = pH10.5
G	66.91	86.45
X	4.88	19.87
C	172.79	30.23
F	30.36	33.50
C x F	33.12	26.75
G x X	not sig.	21.50
Residual	4.65 (10 deg freedom)	3.82 (9 deg freedom)

Analyzing the four factor experiments at pH = 8.5, only the C x F interaction is significant. At a pH of 10.5, the significant interactions are C x F and G x X. Combining the nonsignificant first order interactions with the residual, gives the results shown in Table XX.

At a pH of 8.5 only the grinding variable is significant; the main effects xanthate, cyanide and frother are not significant. However, the interaction cyanide and frother is significant. Therefore, at a pH of 8.5 the following tests can be added, resulting in Table XXI.

At a pH of 10.5, none of the main effects are significant; only the two first order interactions, C x F and G x X. Since these four variables are involved, and the other variable, P, is held constant at P_2 level, each test is significantly different from each other, at a pH of 10.5. No tests at a pH of 10.5 can be added and averaged. Each test represents a random sample of the results using the physical variables listed.

Test No. 22, Table II, yielding a 4.72 pct concentrate grade, was repeated four times, averaging 5.90 pct Cu with a standard deviation of 1.27 pct. Test No. 32, Table II, yielding a 12.66 pct concentrate grade, was repeated four times, averaging 11.68 pct Cu with a standard deviation of 0.97 pct. These two examples show that the conclusions arrived at by factorial design are valid because test No. 22 with a 5.90±1.27 result is certainly different than test No. 32 with an 11.68 ± 0.97 result, and this conclusion with factorial design was arrived at with two tests. The experimental error is taken into consideration in the conclusion. In conventional testing, two tests yielding 4.72 pct and 12.66 pct respectively, may or may not represent differences caused by known variables. While in the laboratory unknown variables are usually kept to a minimum, particularly ore-variability and operator variability, this is almost impossible in plant tests.

Table XXI. Conclusions for Concentrate Grade at pH8.5

Conclusion	Significant Variables: Grind, Mesh	Significant Variables: Cyanide, Lb per Ton	Frother, Lb per Ton	Test No. Table II	Average	Standard Deviation
1	65	0.0	0.05	1,3	5.92	0.27
2	200	0.0	0.05	5,7	2.81	0.86
3	65	0.0	0.10	17,19	5.56	1.00
4	200	0.0	0.10	21,23	2.93	0.58
5	65	0.10	0.05	9,11	8.95	2.05
6	200	0.10	0.05	13,15	7.17	1.82
7	65	0.10	0.10	25,27	18.11	0.90
8	200	0.10	0.10	29,31	9.27	1.13

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Twisted Return Runs for Conveyor Belts

by J. W. Snavely

WITH all the advantages of handling bulk materials by means of belt conveyor also go some problems, one of the most persistent being that of cleaning. When sticky materials are being carried; the build-up of material on the return idler rolls results in difficulty of belt training. Much attention has been given to the problem of cleaning conveyor belts, and a great variety of cleaning devices have been developed. Even the best involve troublesome maintenance, and none can completely remove the fine particles imbedded in the belt cover, which cause rapid wear of the return idler rolls, and at the same time, of the belt cover as well.

One of the major rubber companies has been promoting a two-way belt system, in which the conveyor belt is given two successive 90° twists at each end to enable it to carry material in both directions simultaneously. For many years the flat belt transmission industry has installed quarter-turn and half-turn twists in countless numbers of instances.

While quarter-turn and half-turn twists in transmission belts is a familiar application, the 180° twist apparently has never been previously attempted with a conveyor belt. About a year ago, two officials of the National Iron Co. of Duluth, Lester and Lewis Erickson, proposed twisting the return run of a conveyor belt on an installation that they were designing for one of the major iron ore producers. Since then the soundness of the idea has been demonstrated, both in theory and by practical test, with the result that the installation of two conveyor belts involving the twisting of the return run is now under way.

These two installations are designed to have the return run of the conveyor belt twisted 180° as it leaves the snub pulley at the head drive. The clean underside of the belt is thus placed against the idlers on the return run as well as on the carrying run. Just before it enters the tail pulley, the belt will be twisted an additional 180°, restoring it to its normal position.

Because this twisting of the return run of a conveyor belt is a radical departure from accepted practice, an elaborate and extensive test was conducted early in 1950 to demonstrate that this twisting of the return run could be done successfully, also to establish application data for accomplishing this twisting, and to determine if any special equipment would be required. In studying this concept of twisting the return run of a conveyor belt, a number of problems need to be solved, primarily the ones brought about by deliberately introducing an unequal distribution of stress across the conveyor belt and controlling that maldistribution of stress, while confining it to the return run portion.

The tension conditions existing in the return run of a conveyor belt are clear to all designers. First, the return run carries the initial or slack side tension of the conveyor belt, the tension that must be supplied to the return run to provide proper frictional contact between the belt and the driving pulley so that the necessary power can be transmitted from the driving pulley to the belt without slippage. This slack side tension is supplied to the belt by means of takeups, either of the gravity type, which can be vertical or horizontal, or by means of the screw type.

With inclined or declined belt conveyors the slope tension also must be considered, which is the tension imposed by the weight of the belt hanging from the top pulley. This slope tension frequently can furnish part or even all of the initial tension required. The maximum value of the slope tension will be at the top pulley, and it decreases in direct proportion to the length.

In addition to the foregoing, it frequently is desirable to impose arbitrarily additional slack side tension to provide sufficient tension at the loading point at the tail, so that the belt will adequately support its load between the carrying idlers.

Design Conditions for Twisting

A number of design conditions exist, which must be satisfied successfully to accomplish the twisting of the return run without exceeding normal working limits in any portion of the conveyor belt. It is obvious that the belt edge, in its relation to the center of the belt, must stretch in making a twist, because as the twist is accomplished, the belt edge travels through a longer path than does the center of the belt.

It is further obvious that if the edge of the belt is stretched, a redistribution of stress in the belt is required to allow this edge stretching. Moreover, this stress will be unequal across the width of the belt, having a maximum value at the edges, with a minimum value at the center of the belt.

With correct initial tension in the return run of a conveyor belt, the existing slack side tension will be unequally distributed when a twist is introduced. A condition then exists in which the edge stresses,

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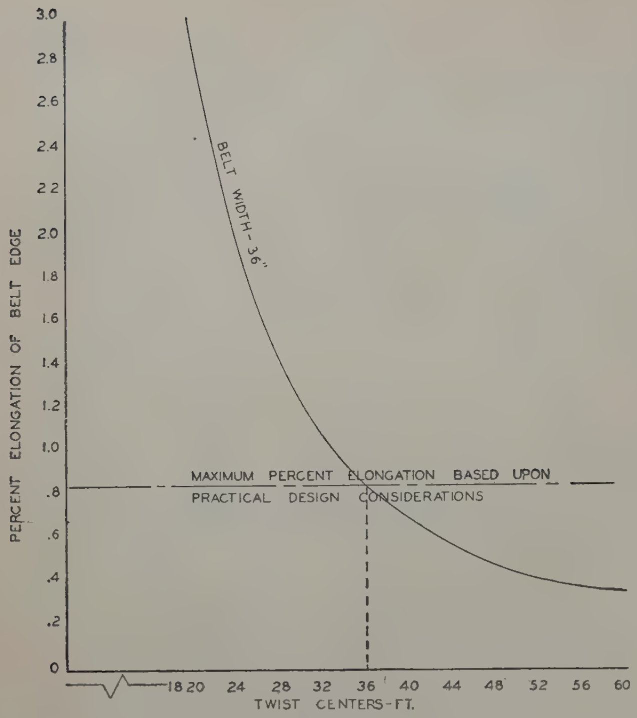


Fig. 1—Tentative minimum twist centers selection chart.

in pounds per inch ply, will be higher than the initial stresses normally uniform across the belt width, and the central stresses will be lower.

Further, when the unequal distribution of stress caused by twisting produces a minimum stress in the central portion of the belt, which is zero or less, the belt has a tendency to *cup* or collapse throughout the twist length. Consequently, to impart a stable condition to the belt in making the twist, tension sometimes must be added to the return run, over and above normal requirements.

Further study shows that the minimum initial tension required to elevate the minimum stress portion of the belt to zero and above is dependent upon the belt width, twist distance, belt elastic modulus, and the number of belt plies. It is necessary to know the elastic modulus of the belt, and it also is obvious that the belt width affects the design because of the longer path that the belt edge of the wider belts travels in negotiating the same twist. By the same token, the twist centers also directly affect the percentage of elongation of the belt edges in relation to the center.

Design Problems

As a matter of practical expediency, existing belt constructions must be used rather than attempt to set up specifications for a special belt construction. In working out the application of twisting the return run of a conveyor belt, it becomes necessary to determine several design components.

1—The twist centers must be established for a given belt width to hold the elongation of the belt edge with relation to the belt center within the working limits of belt construction used, while twisting will usually be carried out in the horizontal plane; doing so in the vertical plane is also practical and both conditions must be studied. From a practical standpoint it is desirable to accomplish the twist in a minimum distance.

2—It must be determined that the normal required initial tension obtained from standard design practices allows a positive stress in the central portion of the belt for the particular belt characteristics and twist centers selected to insure that the belt will be in a stable condition. If necessary, additional tension must be added to meet this requirement.

3—The belt edge stress must be checked to insure that it is below the working limits of the belt construction used.

To show clearly what must be done, appropriate formulae and sample computations used in working out a sample problem involving a 36-in. belt have been prepared. This is the same belt width used in the tests which will be described later. The following is a sample problem which will be used, assuming for illustrative purposes only, that one twist will be in a vertical plane and the other in a horizontal plane.

Stockpile belt to handle iron ore with a tripper on horizontal run. Material, iron ore, 150 lb per cu ft; Capacity, 1000 tons per hr; Belt speed, 400 ft per min; Centers, 600 ft 0 in. horizontal, 40 ft 0 in. vertical; Belt specifications, 36 in. 6 ply, 36 oz duck with ¼-in. and 1/16-in. covers, weighing approx. 12 lb per ft.

Normal Design Tensions, lb:

Effective tension, $et = 6110$
Initial tension, $t = \underline{2020}$
Total tension, $TT = 8130$

Tension at foot, $T_b = 1800$
Slack side tension at head, $T_s = 2020$

Normally, the arrangement of the mechanical equipment for this example would indicate that twists in the horizontal plane would be employed at both the head and tail. However, for illustrative purposes only, a vertical twist will be assumed at the tail and horizontal twist at the head. For purposes of facilitating the illustrative calculations, the belt elastic modulus used will be 2170.

Twist Centers

The selection of twist centers must keep the amount of stretch imposed upon the belt edge in relation to the center, as measured by the percentage of elongation, within the ability of the belt to resist cupping or collapse. As will be shown, a workable rule of thumb for the initial selection of twist centers is to use a twist distance in feet equal to the belt width in inches. Arrived at independently, it confirms the practice that has been successfully used in the transmission belting industry for years.

Based upon these primary considerations, the tentative twist centers for the illustrative problem is selected from Fig. 1. The path transcribed by the belt edge is that of a helix and can be determined by the triangular method. The edge of the belt is the hypotenuse of the triangle; the longer leg, the length between the pulleys; and the shorter side, the circular arc distance transcribed by a point on the edge of the belt in twisting 180°.

The formula which can be used to determine the length of the belt edge for a particular belt width and twist centers is as follows:

Length of Belt Edge in Twisting 180°: Using the following symbols: W = width of belt, in., L = twist centers, in.

$$\text{Length of Belt Edge} - \sqrt{\left(\frac{\pi W}{2}\right)^2 + L^2}$$

From example: Belt width, 36 in.; Tentative twist centers from Fig. 1—36 ft 0 in.

$$\text{Length of Belt Edge} = \sqrt{\left(\frac{\pi 36}{2}\right)^2 + (36 \times 12)^2}$$

$$= 435.68 \text{ in.}$$

The percentage of belt edge elongation with respect to the center is shown in the following formula, clearly indicating that higher stresses will be present in the edge:

$$\frac{\sqrt{\left(\frac{\pi W}{2}\right)^2 + L^2} - L}{L} \times 100$$

$$= \frac{\sqrt{\left(\frac{\pi 36}{2}\right)^2 + (36 \times 12)^2} - (36 \times 12)}{36 \times 12} \times 100$$

$$= 0.852 \text{ pct}$$

Twist Tension

As previously outlined, the return belt must be in a stable condition throughout the twist length, requiring that the minimum stress in the central portion of the belt be zero or above. Therefore, it must be determined that the normal required initial tension obtained from standard design practices, together with the particular belt and twist length selected, permits or allows this zero or positive stress in the central portion of the belt throughout the twist. Theoretically, the minimum stress at the central portion of the belt can be zero, but again, a factor of safety is desirable to positively insure the desired stable condition. It is advisable therefore, that this minimum stress be equal to or greater than 0.2 pct times the elastic modulus of the belt. This value is an arbitrary one and conceivably could be varied. However, it is reasonable in magnitude as indicated by the test conducted.

It is convenient to divide the minimum required twist tension into two parts: 1—the tension required to elevate the minimum stress to zero, and 2—the tension required to elevate the minimum stress to insure a stable belt section. Fig. 2 illustrates point 1 and the following formula shows the derivation of point 2 together with the additional slack side tension to be added to the normal tension to arrive at the minimum required twist tension.

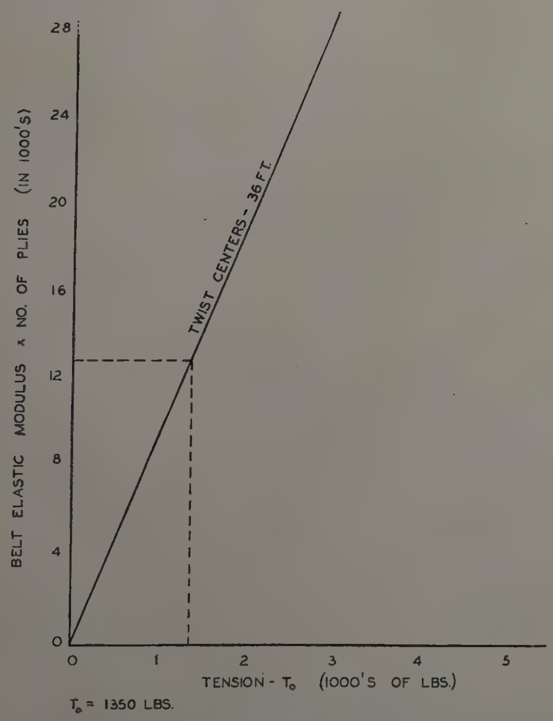


Fig. 2—Tension required to elevate minimum stress to zero.

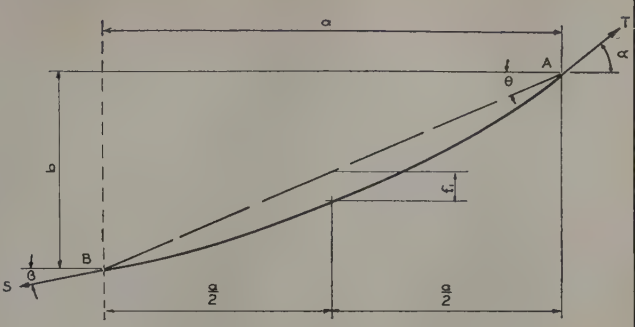


Fig 3—Theoretical sag.

Using the following symbols: a = Horizontal distance between points of belt supports, ft; b = Vertical distance between points of belt supports, ft; θ = Angle, which AB makes with the horizontal, referred to as angle of inclination of twist $(= \tan^{-1} \frac{b}{a})$; T = Twist tension at head which is equal to new T_s; α = Approach angle for centerline of belt at upper point of suspension-degrees; β = Approach angle for centerline of belt at lower point of suspension-degrees, f_1 = Theoretical sag, ft; $n_1 = \frac{f_1}{a}$ = sag ratio; w = wt of belt, lb per ft.

$$\begin{aligned} T &= T_s + \Delta T \\ &= 2020 + 488 \\ &= 2508 \text{ lb} \end{aligned}$$

$$T = \tfrac{1}{2}\, wa \left[\frac{1}{16 n_1^2} + \frac{\text{sine } \theta}{2 n_1} + 1\right]^{1/2}$$

$$2508 = \tfrac{1}{2}\,(12)\quad(36)\left[\frac{1}{16\left(\frac{f_1}{36}\right)^2} + \frac{\text{sine } 8^\circ}{2\left(\frac{f_1}{36}\right)} + 1\right]^{1/2}.$$

$f_1 = 0.791$ ft = 0 ft — 9½ in.

Using the following symbols: T = Minimum required twist tension, lb, T_m = Tension required to elevate minimum stress to insure a stable belt section throughout the twist, lb, T_o = Tension required to elevate minimum stress to zero, lb (obtained from Fig. 2), ΔT = Additional slack side tension to be added, lb, t_m = Minimum stress to insure a stable belt section, lb per in. ply, W = Width of belt, in., P = Number of plies, T_b = Tension at foot, lb, t_m = Belt elastic modulus × 0.2 pct.

$$\begin{aligned} T_m &= t_m\, WP \\ &= 0.2 \text{ pct } (2170)\ (36)\ (6) \\ &= 938 \text{ lb} \end{aligned}$$

$$\begin{aligned} T &= T_m + T_o \\ &= 938 + 1350 \\ &= 2288 \text{ lb which becomes the new } T_b \end{aligned}$$

$$\begin{aligned} \Delta T &= T - T_b \\ &= 2288 - 1800 \\ &= 488 \text{ lb} \end{aligned}$$

It has been shown that the belt edge stress will be elevated above the normal initial belt stresses because of the twist. It is necessary therefore to check the belt edge stress to insure that it is below the working limits of the belt used as shown by the fol-

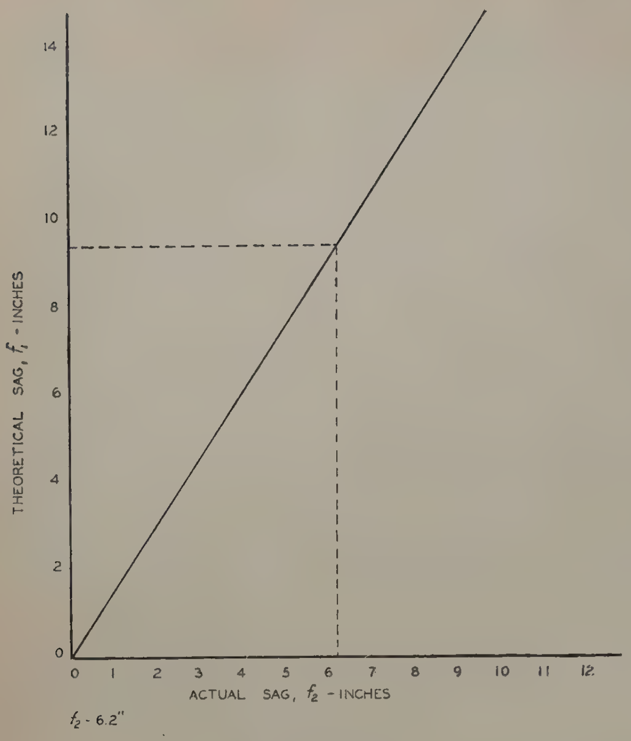


Fig. 4—Actual sag.

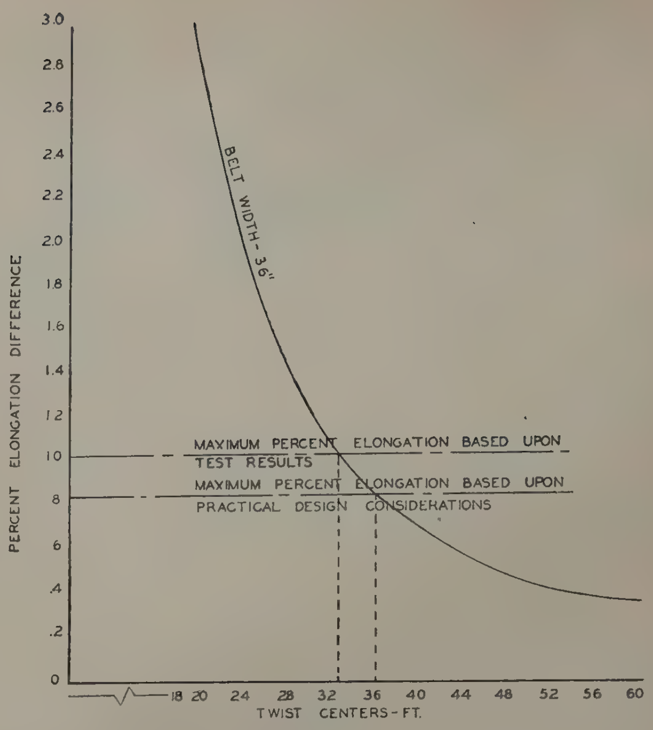


Fig. 5—Tentative minimum twist centers selection chart.

lowing formula. Examination reveals that the belt elastic modulus becomes a very important consideration in the selection of belts to satisfy this requirement.

Belt Edge Stress (Vertical Twist), Using the following symbols: T = New T_b, twist tension, lb, P = Number of belt plies, W = Width of belt, in., t_s = Maximum stress at edge of belt, lb per in. ply, t_m = Minimum stress at center of belt, lb per in. ply.

$$T = PW\ t_m + \frac{PW}{3}\ (t_s - t_m)$$

$$2288 = 6\ (36)\ (4.34) + \frac{6\ (36)}{3}\ (t_s - 4.34)$$

$$t_s = 23.12 \text{ lb per in. ply}$$

Horizontal Twists

It can be appreciated that the foregoing is applicable to a vertical twist because the distribution of stress is symmetrical about the centerline; that is, the stresses of both edges will be equal. However, twists completed in a horizontal plane are acted upon by the external force of gravity, requiring further analysis.

A conveyor belt is flexible to a certain degree, and for that reason the curve which the belt transcribes between two pulleys when suspended in a horizontal plane approaches that of a catenary in form. The end result is that the belt edges stretch unequally, causing a difference in belt edge stresses. The solution of the highly complex exponential form of catenary equations is extremely difficult, and since the results for low sag ratios, where the sag is less than 10 pct of the span, are for all practical purposes the same, the solution used follows the parabolic form of suspension.

However, the belt does possess some longitudinal rigidity, because of its cross-section in the twist, the magnitude of which is dependent upon its width and construction. The amount of actual sag of the belt between the twist pulleys therefore will be appreciably less than the theoretical amount of sag if the belt were treated as a completely flexible member.

It should be clear that the amount of actual sag determines the maldistribution of stress across the width of the conveyor belt. As mentioned above, the determination of the theoretical amount of sag is that of a parabolic function, and Fig. 3 shows the formula and how it is applied to the example used.

The amount of difference between theoretical and actual sag is largely a function of the moment of inertia of the belt about its horizontal axis at any specific point of its rotation throughout the twist, together with the belt characteristics. For the 36-in. belt and the construction used in the test, the relationship of the actual sag to the theoretical sag was 0.652, and Fig. 4 shows this relationship.

In the twisting of the return belt in the horizontal plane, one of the critical problems is to limit the stress in the high tension edge of the belt. After the sag calculations are made, it is necessary to check that the stress in the high tension edge be below the recommended maximum rating of the belt. As the maldistribution is dependent upon the actual amount of sag, the effect of this sag must be considered. The mathematics involved become somewhat complicated but are best worked out knowing that the stress distribution curve across the width of the belt follows that of a parabola. The formula used, and its application to the example are illustrated below:

Critical Belt Edge Stress (Horizontal Twist): Using the following symbols: X = Distance from low tension edge to minimum stress, in., Y = Distance from high tension edge to minimum stress, in., t_m = Minimum stress, lb per in. ply, t_{sH} = Stress at high tension edge, lb per in. ply, t_{sL} = Stress at low tension edge, lb per in. ply, f_2 = Actual sag, in., W = Width of belt, in., T = Twist tension at head, lb, E = Belt elastic modulus, T_o = Tension required to elevate minimum stress to zero, lb, T_m = Tension required to elevate minimum stress across width of belt, lb.

$$Y = \frac{W}{2} + f_2 = \frac{36}{2} + 6.2 = 24.2 \text{ in.}$$

$$X = \frac{W}{2} - f_2 = \frac{36}{2} - 6.2 \quad 11.8 \text{ in.}$$

$$T_m = T - T_o = 2508 - 1350 = 1158 \text{ lb}$$

$$t_m = \frac{T_m}{WP} = \frac{1158}{36(6)} = 5.36 \text{ lb per in. ply}$$

$$T_o = \frac{P(X^2 + Y^2)\ (t_{sH} - t_m)}{3\,Y}$$

$$1350 = \frac{6(11.8^2 + 24.2^2)\ (t_{sH} - 5.36)}{3(24.2)}$$

$$t_{sH} = 27.95 \text{ lb per in. ply}$$

For a 36-in. belt, the test results showed that the minimum twist centers in which the belt could be twisted, analyzed from the standpoint of tendency to collapse because of the uneven distribution of stress, was 1.04 pct elongation of the edge with respect to the center. This was correlated to theoretical minimum twist centers and substantiated by test results.

Sound engineering practice dictates the use of a factor of safety in design work such as this, and a commonly applied factor is 80 pct of theoretical value. Eighty pct of the 1.04 pct of elongation difference gives a working value of 0.832 pct permissible elongation difference. For convenience, this is slightly increased to 0.852 pct, because at that figure, the twist centers in feet are exactly equal to the belt width in inches.

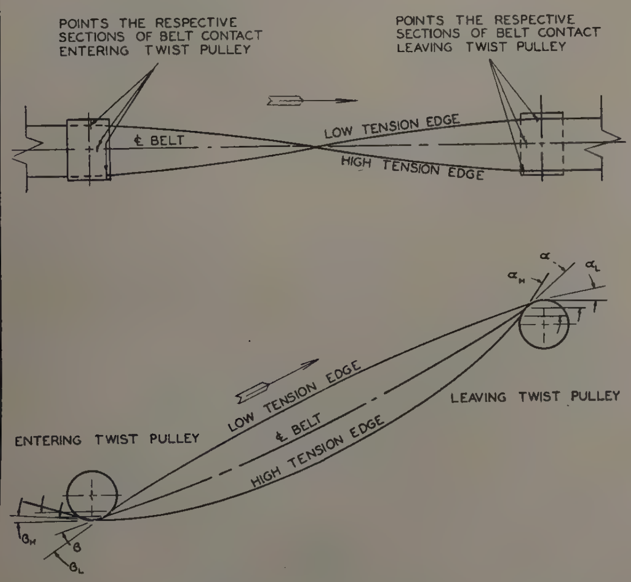

POINTS THE RESPECTIVE SECTIONS OF BELT CONTACT ENTERING TWIST PULLEY
POINTS THE RESPECTIVE SECTIONS OF BELT CONTACT LEAVING TWIST PULLEY
℄ BELT
LOW TENSION EDGE
HIGH TENSION EDGE
α
α_H
α_L
LOW TENSION EDGE
℄ BELT
HIGH TENSION EDGE
ENTERING TWIST PULLEY
LEAVING TWIST PULLEY
β_H
β
β_L

Fig. 6—Approach angles.

The slope angle of the belt at the points of support measured with respect to the horizontal is termed "approach angle."

Leaving Twist Pulley	Entering Twist Pulley
α — Centerline of belt	β — Centerline of belt
α_L — Low tension edge	β_L — Low tension edge
α_H — High tension edge	β_H — High tension edge

Using the following symbols:

f = Theoretical sag, ft

a = Horizontal distance between points of belt supports, ft

b = Vertical distance between points of belt supports, ft

$$\tan\alpha = \frac{b + 4f_1}{a} = \frac{5.05 + 4(0.791)}{36} = 0.228$$

$$\alpha = 12^\circ - 50'$$

$$\tan\beta = \frac{b - 4f_1}{a} = \frac{5.05 - 4(0.791)}{36} = 0.0524$$

$$\beta = 3^\circ - 0'$$

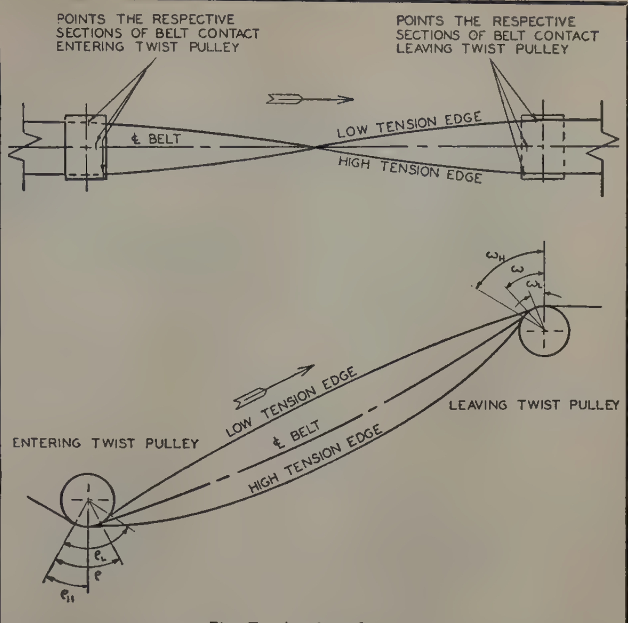

POINTS THE RESPECTIVE SECTIONS OF BELT CONTACT ENTERING TWIST PULLEY
POINTS THE RESPECTIVE SECTIONS OF BELT CONTACT LEAVING TWIST PULLEY
℄ BELT
LOW TENSION EDGE
HIGH TENSION EDGE
ω_H
ω
ω_L
LOW TENSION EDGE
℄ BELT
HIGH TENSION EDGE
ENTERING TWIST PULLEY
LEAVING TWIST PULLEY
ρ_L
ρ
ρ_H

Fig. 7—Angles of wrap.

The angular measurement of belt contact on the twist pulleys is termed "angle of wrap."

Leaving Twist Pulley —

ω — Centerline of belt

ω_L — Low tension edge

ω_H — High tension edge

Entering Twist Pulley —

ρ — Centerline of belt

ρ_L — Low tension edge

ρ_H — High tension edge

Fig. 5 has been prepared to show the percentage of elongation curve for the 36-in. belt, together with the limiting design percent of 0.852. This figure actually is an overlay of Fig. 1 with the minimum percent elongation of 1.04 sketched in for comparative purposes.

In analyzing what takes place during the twisting of the return belt, the term approach angle must be used. In leaving and entering the twist pulley, the conveyor belt does not contact the twist pulley in the same plane. Going into the twist, the belt has a substantially uniform distribution of stress across its width, and the approach angles of the edges vary equally with the approach angle of the center of the belt. This variation is largely independent of the centers of the twist and the amount of sag and tension but can be more closely correlated to a function of width and transverse rigidity of the belt.

In leaving the twist, the belt has an uneven distribution of stress, imposed by the twisting, and the approach angles of the edges vary unequally from the approach angle at the center of the belt, with one edge of the belt having a greater variation than the other. In general, however, the pattern of a helical generation is followed.

Fig. 6 illustrates what is meant by these approach angles.

It is essential that there be sufficient contact between the return belt and the twist pulleys to permit these twist pulleys to influence and train the belt. This amount of contact is commonly referred to as angle of wrap, and in accomplishing the twisting of the return belt, this is most critical on the leaving twist pulley, where a minimum of 5° of wrap is required at the minimum stressed edge.

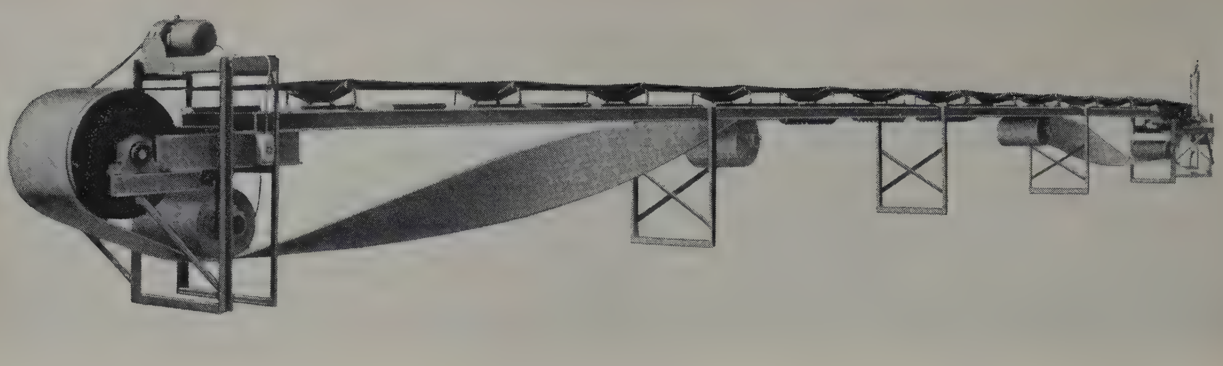


Fig. 8—Test conveyor.

For a given twist centers and belt weight, the angle of wrap is dependent upon the approach angles and the relative position of the belt travel with relation to the pulleys. Fig. 7 illustrates these angles of wrap.

Largely based upon analysis of the test results, the following comments on the various equipment items seem appropriate. Generally speaking, no special equipment is going to be required to twist successfully the return run of a conveyor belt.

It should be possible to accomplish successfully the twisting of the return run of conveyor belts using standard fabric constructions of belting and some low modulus cord constructions as well. The final determination of the twist centers necessarily will need to take into consideration the construction of the belting used, and it is essential that the elastic modulus of the belt, within reasonable limits, be obtained from the belting supplier.

Until a number of successful applications have confirmed the design procedure, it will be essential to have the belting supplier also make a simple static twist test with a sample piece of the actual belt to be supplied. This is not particularly difficult or expensive to set up, and will serve as a final important check on the design work that has been done.

Mechanical Equipment

The pulleys used in accomplishing the twist normally would be referred to as bend pulleys, but because of their particular purpose, they have been and will be referred to as twist pulleys. Because of the uneven distribution of stress across the width of the belt in accomplishing the twist, flat-faced pulleys within normal tolerances of manufacture are necessary. Crowned pulleys would only add to the amount of adjustment necessary in belt training. Also, the pulleys should be slightly wider than normally used, about 4 in. wider than the belt up to widths of 36 in., and 6 in. wider than the belt for widths beyond 36 in. This additional width to the pulley face is merely to safeguard the edges of the belt should alignment difficulty develop during initial run-in period.

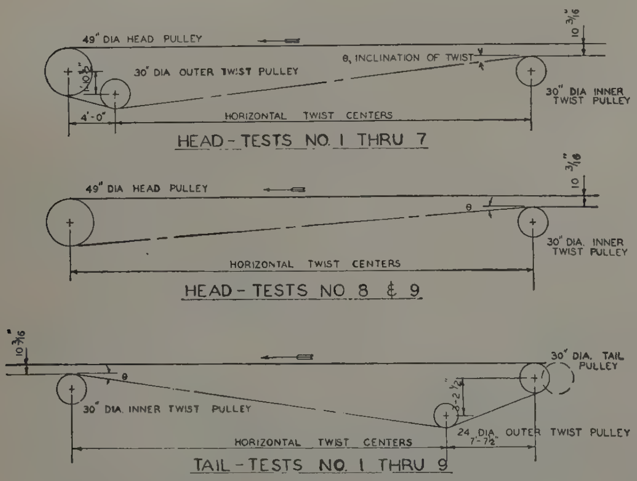


Fig. 9—Schematic arrangement of test conveyor.

It is the leaving pulley of each twist that is the critical one, and the only pulley on which adjustment need be made. Such adjustment can be made entirely in the horizontal plane, and based upon the experience of the test, it is not anticipated that such adjustment should ever need to exceed 3 in. Small adjustments of 1/8 in. to 1/4 in. produced a considerable belt training effect. Therefore, it would be recommended that an adjustable type of base plate, equipped with adjusting screw and locknuts be provided, and antifriction, self-aligning bearings used for the twist pulley mountings.

Any reasonable arrangement of pulleys can be used to complete the twist in satisfying the requirements of a specific installation problem, provided the necessary design considerations are carried out. It is emphasized that a twist pulley with adjustment facilities always be provided preceding the tail pulley, for central loading on the belt.

Idlers: Conventional idler equipment will be completely adequate, and there would seem to be no need for the use of training return idlers beyond normal requirements because of the very effective training influence of the leaving twist pulleys. The need for training idlers on the return run of the conveyor belt should be sharply decreased, because there will be no build-up of material on the return rolls tending to deflect the belt to one side or the other. The need for expensive special constructions of return idlers also disappears, and the conventional solid roll return idler can be used with complete assurance, with a much longer life expectancy than ever heretofore possible.

Takeups: It is obvious that very accurate control of tensions must be maintained at all times, and it therefore becomes essential that an automatic type of takeup be used to insure the proper amount of slack side tension at all times and under all conditions. This is best accomplished with the gravity type of takeup, either horizontal or vertical, and it further becomes important to see that the takeup is

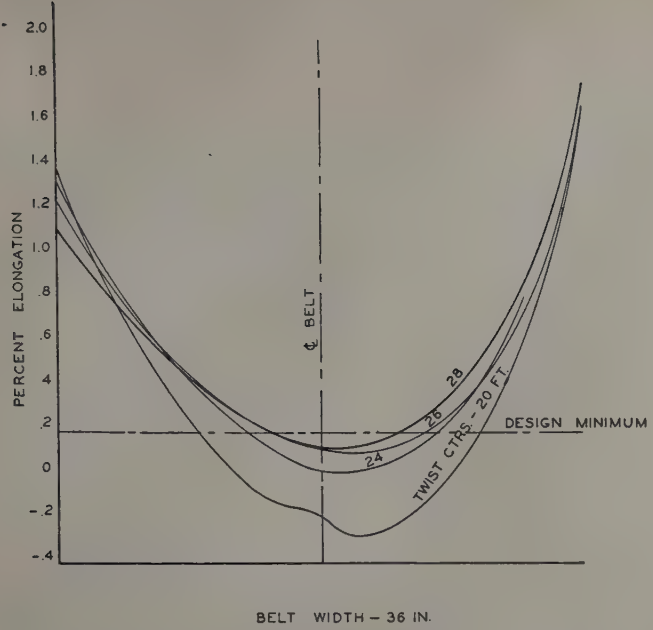


Fig. 10—Longitudinal elongation across width of belt, test results for short twist centers.

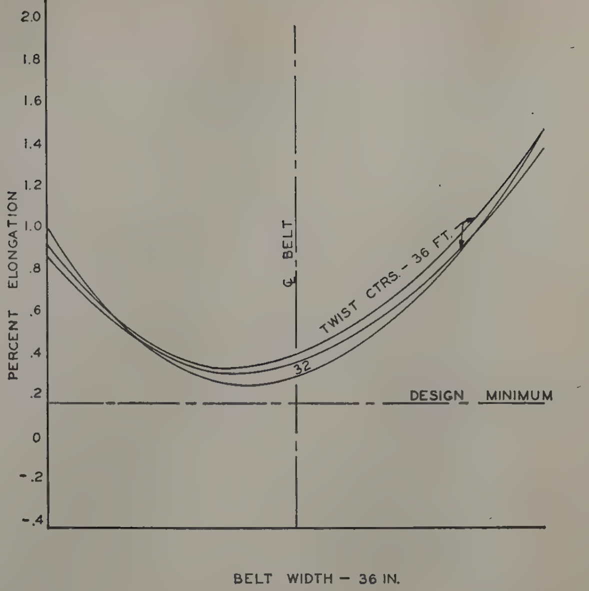


Fig. 11—Longitudinal elongation across width of belt, test results for more desirable twist centers.

free and operative at all times. It is desirable to have the takeup location where it is most sensitive, which is normally following the head pulley.

Drives and Controls: The use of customary drives and controls is completely satisfactory. A greater degree of control of accelerating tension becomes highly desirable to limit any surges of tension that might be sent through the conveyor belt, which could occasionally cause momentary fleeting of the belt on the twist pulleys.

Operating Techniques

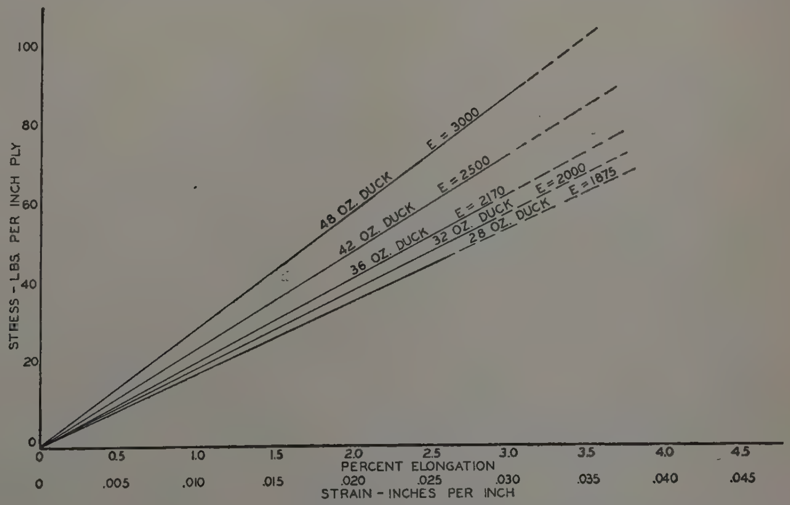
A number of operating techniques became apparent from the test as it was conducted. The most important concerns the manner of introducing the twist in the belt. It is essential that the twist introduced into the return run of the conveyor belt be continued in the same direction so as to accomplish a total of 360°. If the belt is twisted 180° and then returned 180°, the same belt edge is being kept at maximum tension, which will have two results. First, the belt will be much more difficult to train, since it would appear that there is some buildup of residual stress that passes beyond the twist pulleys themselves. In addition, and more serious, over a period of time the belt would take permanent stretch to a greater degree along the continuous high tension edge than it would on the other edge, and successful belt training would then become virtually an impossibility.

It is also necessary for belts employing a twisted return run that only vulcanized splices be used. Not only is the mechanical splice, regardless of the type used, not as flexible as the belt itself, but it likewise does not distribute the stress across the belt in the normal pattern. Moreover, the mechanical splice will not permit the use of maximum tension ratings in applying the conveyor belt, and they are also particularly weak at the belt edge, which for twisting is the location at which they should be the strongest.

As to belt training, it is the leaving pulley that trains the belt, and the belt tends to run off on the low tension edge. The adjustment to train the belt can be made by moving the pulleys in a horizontal plane, moving the low tension edge of the pulley forward in the direction of belt travel. The belt is quite sensitive to such pulley adjustment, and a very small amount of movement immediately imposes the required training effect.

It would appear that speed has little influence on the belt operation, as the increase or decrease of

Fig. 12—Stress-strain characteristic curves of conveyor belt fabrics.



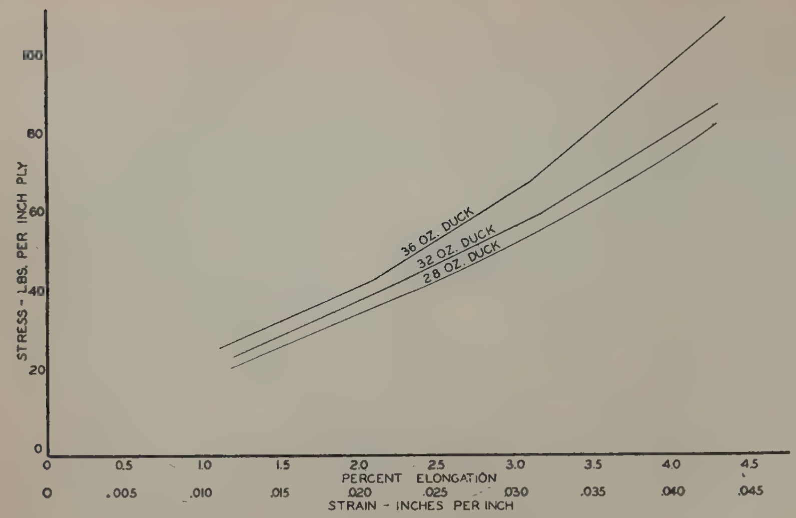


Fig. 13 — Stress-strain characteristic curves of conveyor belt fabrics, typical test data furnished by conveyor belt manufacturer B.

speed in the test conducted imposed no appreciable effect.

Acceleration control does become relatively important, as across-the-line starting places a surge of tension in the system. When the takeup is located so that it cannot react immediately to maintain a constant tension, there obviously would be a momentary decrease in the tension in the twist following the head pulley, and a decrease in slack side tension would tend to cause the return run to momentarily fleet on the twist pulley, being corrected when the conveyor is up to speed and the takeup counterweight has reacted.

To minimize this action, it is recommended that starting controls be used to limit the accelerating tension to approximately 135 pct of normal operating tension, and that the takeup be placed at the point of most sensitive reaction.

Tests and Results

The test that was conducted, and its confirmation of the mathematical basis of design is of considerable interest. The test program was a cooperative development project carried on by the National Iron Co. of Duluth, Minn., the Mining Department of Cleveland-Cliffs Iron Co. of Ishpeming, Mich., the Flat Belting Field Engineering & Development Div. of the B. F. Goodrich Co. of Akron, Ohio, and the Conveyor and Process Equipment Div. of the Chain Belt Co. The tests were performed May 10 to May 13, 1950, at the National Iron Co.

Fig. 8 shows the test conveyor setup which clearly shows the twisting at both the head and tail. Fig. 9 is a presentation of the schematic arrangement of the types of twisting employed in the conducting of the tests in question.

The following are the specifications of the test conveyor: *Belt*—Goodrich Longlife, 36 in., 5 ply, 32 oz duck with ⅛ x 1/16 in. covers, 2500 to 3000 psi tensile, 16 to 19 lb friction, belt weight 8 lb per ft (actual weight). *Belt speed*—271 and 526 ft per min. *Conveyor centers*—130 ft 1½ in. horizontal. *Takeup*—horizontal gravity takeup on tail pulley, counterweight weighed and variable. *Idlers*—REX style No. 32 troughing idlers with 5 in. diam rolls, inclined 2° in direction of belt travel at 10 ft spacing, REX style No. 55RC—return idlers at 10 ft spacing. *Drive*—Gearmotor, Westinghouse, 10 hp class I, 220/440 v ac motor, output rpm 84, 26/13 amp, ratio 20.8-1, 1750 rpm, unit No. CJA 324, 60 cycle; chain drive, No. 100 REX roller drive chain; 18 T, 7.198 in. P.D. driver for 271 ft per min; 70 T, 27.861 in. P.D.

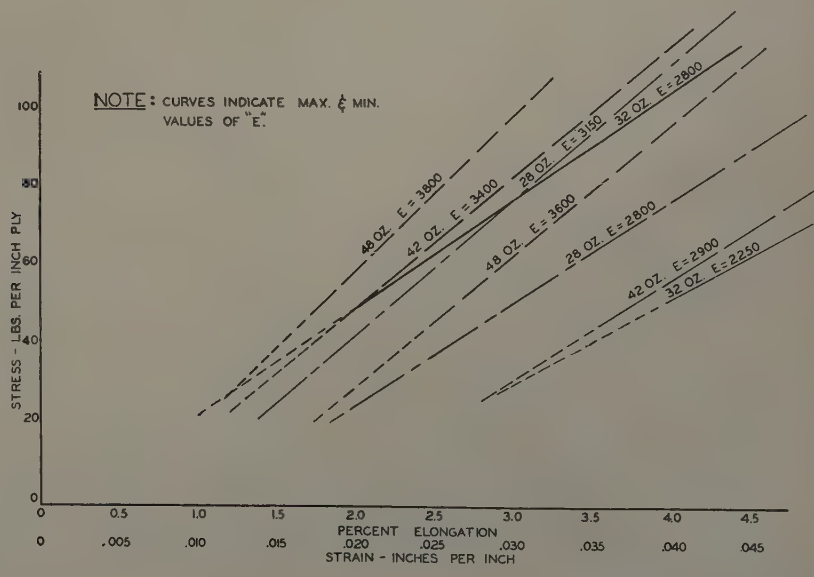


Fig. 14 — Stress-strain characteristic curves of conveyor belt fabrics, representative test data furnished by conveyor belt manufacturer A.

Table I. Test Data

				Twist Tension				Approach Angles, Deg			Angles of Wrap, Deg		
Test No.	Twist	Length of Twist	Inclination of Twist	Lb	Lb, In. Ply	Sag, In.	Twist Pulley	High Tension Edge	Center	Low Tension Edge	High Tension Edge	Center	Low Tension Edge
7	Head	32 ft, 3¾ in.	7°-58′	1800	10	4 3/16	Outer	— 3	5	13	19	28	33
							Inner	17	12½	2	19	15	5
8	Head	36 ft, 1 11/16 in.	5°-05′	1800	10	6 1/16	Head	— 6	0	8	174	180	188
							Inner	14	9	1	16	12	3
9	Head	36 ft, 1 11/16 in.	5′-05′	1800	10	5⅝	Head	— 6	0	7	174	180	187
							Inner	14	10	1	13	13	3

driven; 35 T, 13.945 in. P.D. driver for 526 ft per min. *Pulleys,* head, 48 in. diam x 40 in. face with ½ in. thick vulcanized chevron type rubber lagging; tail, 30 in. diam x 40 in. face; head outside twist pulley, 30 in. diam x 40 in. face; head inside twist pulley, 30 in. diam x 40 in. face; tail outside twist pulley, 24 in. diam x 40 in. face; tail inside twist pulley, 30 in. diam x 40 in. face.

All of the pulleys are of the solid welded steel construction with straight face with the exception of the tail outside twist pulley, which could be classified as crowned, or an irregular flat face pulley.

After the conveyor was set up, the procedure following was used: The belt was painted with a rubber base Kemtone paint in three sections 6 ft apart. These were then scribed with longitudinal lines 33 in. long at 3-in. intervals transversely across the belt. Tension had been released to insure correct length under no stress. The tension was then reapplied and the elongation of the measured sections were tabulated. Readings were taken in the following positions: 1—Between head and outer twist pulley, 2—Three positions through head twist, 3—One position between inner head twist pulley and first return idler, 4—One position between the last idler and inner tail twist pulley, 5—Three positions through tail twist, 6—Between outer tail twist pulley and tail pulley, 7—One position immediately ahead of tail pulley on carrying side, 8—One position immediately behind head pulley on carrying side.

In addition to the strain readings mentioned, additional pertinent data taken consisted of recording the slack side tension, twist centers, slope of twist, approach of belt to the twist pulleys, actual sag, cupping, and angle of wrap on the twist pulleys.

Fig. 10 gives the results of four of the tests in which the twist was introduced through centers that were too short. It is to be noted that in every case the low points of the curves fall below the minimum recommended design figure of 0.2 pct elongation at the minimum stressed portion of the belt, and that two of the curves drop below zero.

Fig. 11 shows a similar presentation of the readings of three of the tests which were conducted at satisfactory twist distances, and it will be noted that in these instances, the low points of the curves are all above the design minimum. Study also reveals how closely the variables involved in the design of twisted belts are interrelated.

The remaining pertinent data for the three tests shown at satisfactory twist distances are given in Table I.

From the previous discussion, it is evident that the characteristics of the belt construction used are an important factor. However, the determination of these characteristics, in other words, the elastic modulus for the belt construction, is not always readily possible to establish to a precise degree. Because of the construction of conveyor belts, and the nature of the materials used, there is bound to be appreciable variation which cannot be avoided. The problem is clearly evident from a comparison of Figs. 12-14 which show the elastic modulus for several belt constructions from three different sources. It is therefore obvious, and it must be emphasized, that the approval of the belting supplier must be obtained to achieve successful belt performance.

Advantages

It must be immediately apparent that the basic advantage is the placing of the clean side of the belt on the return idlers. This will immediately provide freedom from many operating annoyances and specifically will provide such advantages as:

1—No build-up of material on the return idler rolls. Sticky materials and wet materials in freezing temperatures can be handled readily. 2—Much longer return idler life. 3—Increases belt cover life appreciably. 4—Elimination of all dribble under the conveyor, except underneath the twists. 5—Elimination of the need for deck plates except at the head and tail sections of the conveyor.

Disadvantages

There naturally are going to be some inherent disadvantages, although it would appear that these are heavily outweighed. Some of the disadvantages would seem to be:

1—The distance required for twisting makes it impractical for short conveyors. 2—It may be necessary to use slightly heavier back covers on the belts. 3—Additional pulleys are required. 4—Some sacrifice of working tension in the belt. 5—If the mechanical arrangement of pulleys is such that some pulleys are in contact with the dirty side of the belt, cleaning these pulleys may be necessary.

Conclusions

On paper, and from the test, the twisting of the return run of a conveyor belt would appear to have great promise of being an important step forward in the art of belt conveying. It must be remembered however, that the test conducted was of very short duration, and solely with an empty belt.

Good judgment dictates reserving final evaluation until its soundness has been demonstrated by actual and prolonged field operation. During the ensuing months this field experience will be gained, and it is confidently expected that a report a year from now will largely confirm the present promise.

Acknowledgments

Complete credit for the mathematical analysis and derivations presented is hereby given to R. A. Zettel of Chain Belt Co. Acknowledgment is also made of the assent by the members of the development project test for the presentation of this paper.

Fig. 1—Low spot in an acid polluted stream.

The Formation of Acid Mine Drainage

by Kenneth L. Temple

and Arthur R. Colmer

ACID coal mine drainage presents a peculiarly difficult problem for two principal reasons. First is the fact that the amount of acid water discharged from active and abandoned mines constantly increases as coal fields are developed and thereby creates a steadily worsening nuisance of rapidly growing importance to the general public as well as to the coal industry. Second is the even more significant fact that there is no method yet known that will appreciably alleviate the acid water problem except at totally prohibitive cost to the coal operators. The problem is thus seen to be an urgent one. With a growing need for good quality water and a growing public demand for abatement of stream pollution, a large portion of the soft coal industry finds itself in the position of year by year dumping more and more acid into the streams and waterways and having no way in sight of improving the situation or even of keeping it from getting worse.

While some studies have been made, it is only in the last few years that the problem has been seriously considered, chiefly by the group of workers under the direction of S. A. Braley at Mellon Institute and by the Bituminous Coal Research, Inc. fellowship at West Virginia University. Hinkle and Koehler[1] reported the results of the earlier West Virginia work. This work pointed to certain biological aspects which had not been noticed previously. These may be summarized as two separate findings. One was that sulphur oxidizing bacteria were present in acid mine water and that their possible role in contributing to the acidity should be investigated. The other was

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that the oxidation of ferrous iron in acid mine water to produce the characteristic red color took place only under conditions which showed that the process was a biological one and not a simple atmospheric oxidation. Since that time the biological aspects of acid mine water have been studied more intensively, and some of the results obtained constitute the subject of this report.

All acid mine water is formed by the passage of ordinary ground water through a coal mine which may be either active, inactive, or even long abandoned. However a similar product often results from leaching of gob piles. The original ground water is never acid but may be neutral or slightly alkaline. On passing through an acid forming mine the water becomes acid in reaction. It develops a high titratable acidity when measured against a strong base such as caustic. It is found to contain a high content of inorganic salts, particularly ferrous iron and sulphate. Calcium, magnesium, and aluminum levels are also increased. The metallic ions and sulphate ions balance each other so that it may be said that there is no free sulphuric acid, in spite of the actual high acidity. No acid is formed in a coal seam which has not been opened up by mining. All acid water sooner or later undergoes a change in which the ferrous iron Fe^{++} is oxidized to ferric iron Fe^{+++}; the ferric content is responsible for the brown or red color. The quantitative acidity and hardness of acid waters covers all gradations from unaltered ground water to a highly acid water having a pH of 2 to 3, a total titratable acidity of several thousand ppm and an iron content of perhaps 2000 ppm Fe.

Ferrous Iron Oxidation

With regard to the discovery previously mentioned, that the ferrous to ferric change is biological, it may be said that this has been fully substantiated. The reaction is brought about by a bacterium, not hitherto known, which causes the oxidation to proceed in spite of the fact that ferrous iron is stable to atmospheric oxidation at the low pH levels encountered. While much work has been done on this phase of the problem[2] it is not of direct concern to the subject of this paper and will not be further discussed.

Sulphur Oxidizing Bacteria

The sulphur oxidizing bacterium discovered by Colmer and Hinkle[3] turned out to be *Thiobacillus thiooxidans,* a bacterium that has been known and studied quite intensively for many years. For the purposes of this problem the physiological characteristics of the organism are of prime importance. It belongs to a group of bacteria which live upon inorganic matter entirely, and get all of their nutrient requirements therefrom. They grow and reproduce by turning inorganic matter into organic cell material. One of the requirements of all forms of life, just as of any engine, is a fuel, a source of energy. *Thiobacillus thiooxidans* uses inorganic sulphur, the element and its compounds, as fuel. It oxidizes sulphur, or thiosulphate or tetrathionate into sulphate whereby it manufactures sulphuric acid. The question whether disulphides are similarly oxidized has been investigated before but not satisfactorily answered. The most that may be said is that soils or composts containing pyrite develop a large population of *T. thiooxidans* as the acidity develops.

A survey of acid mine waters of the Morgantown area from different coal seams and including a few samples from other parts of the state and from Pennsylvania showed that *T. thiooxidans* is present in all acid mine waters examined and may reasonably be supposed to be in all acid mine waters. The organism is not found in other streams and has never been encountered in nature except where oxidizable sulphur compounds were present. Since it is not believed to grow in any other way than by oxidizing sulphuritic material, it is a logical assumption that *T. thiooxidans* when found in acid water is actively contributing to the acidity of that water.

Upper Freeport Seam

During an investigation of a mine on the Upper Freeport seam more specific evidence of bacterial association with acid formation came to light. This particular mine was in a very early development stage and was not near any other mines. It contained one area that was very wet, most of the mine being dry. The wet area proved to be acid forming and an attempt was made to study the exact location of acid formation. This represented the first opportunity to see acid formation occurring inasmuch as older mines usually have acid formation largely restricted to worked out and inaccessible areas. By a simple pH test using an indicator solution on white filter paper applied to the damp surface of the wall, it was possible to find the source of acid in this mine.

Only a very small portion of the wet area was acid forming. The entire section was characterized by an unusually bad top for the seam, a slickensided shale of loose structure which could not be held in place. In consequence, the height of the entry at that point was almost 7 ft, although the coal measured only 40 in. Work on this entry was stopped, and eventually connection was established by driving back toward it, using both short posts and roof bolts which held the roof in place. The exposed roof strata consist of a soft shale, about a foot of rider coal (much thicker here than is usual) and more shale. Tests for pH indicated that most of the water was essentially neutral and similar to ordinary ground water. This applies to both roof drips and the water on the ribs. In a very few localized spots the water was highly acid. These spots were present in the shale and in the rider coal but not in the Freeport coal proper. These spots when in the rider coal could also be distinguished visually by the deposits of

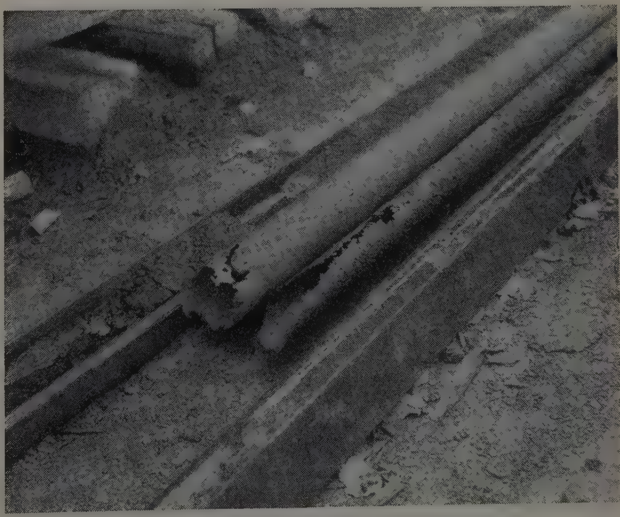

Fig. 2—Typical black iron pipe corroded from inside. These pipes are replaced approximately once a week.

yellow copiapite, a basic hydrated iron sulphate which is an oxidation product of pyrite and is acidic in reaction but has a strong superficial resemblance to elemental sulphur. Acid spots in shale could not be detected visually. There were numerous small pyritic concretions resembling miniature sulphur balls. These did not test acid, and their distribution bore no relation to the location of acid spots. *T. thiooxidans* was found in the rider coal and shales wherever there was acid. It was absent from adjacent neutral areas. Only the surface was acid. When the surface was scraped away, the underlying material was neutral and contained no sulphur oxidizing bacteria. In a few days this newly exposed surface became acid and contained a vigorous population of *T. thiooxidans*.

These observations on the Upper Freeport seam lead one to conclude that the readily oxidizable sulphuritic material is in the roof strata of this seam and is of exceedingly local distribution, accounting for the fact that Upper Freeport water is only moderately acid when the discharge from an entire mine or area is considered. Furthermore the oxidation of the rapidly oxidizable sulphuritic material, whatever its nature may be, is intimately associated with the development of the sulphuric acid manufacturing bacterium *T. thiooxidans.* In addition it is evident that the acid spots contained a sulphuritic material much more readily susceptible to oxidation than the prominent and visibly apparent pyritic concretions. It is of interest in this connection that so far all laboratory experiments with *T. thiooxidans* have been conducted with the larger pyritic concretions and not with the acid forming shales. There is no question but that acid formation in this mine could be stopped if the surfaces of the roof and walls were coated with some substance sealing them against air. This might prove difficult to accomplish.

Pittsburgh Seam

The Pittsburgh coal in the vicinity of Morgantown is of two distinct types as regards acid water production. The two types are separated geographically by the Monongahela River. This distinction coincides with a difference in the roof structure. On the western side of the Monongahela River the Pittsburgh coal has a characteristic roof comprised successively of shale, coal, shale, coal, dense hard shale, and soft shale up to limestone. This is a general picture and, depending on the particular location, these layers may sometimes be subdivided further. At one point in an acid producing mine these strata are 25 to 30 ft thick between the coal and limestone. Above the Pittsburgh the Sewickley seam and sometimes the Redstone seam is present. On the eastern side of the Monongahela River between the Cheat River and Decker's Creek the roof strata consist of a shale and sandstone, with thin cover and no limestone or coal overhead. The shale, moreover, is exceedingly variable and may be entirely absent, or have a thin layer of sandstone between it and the coal. The sandstone is of two types, one yellow, and the other light gray to white.

The sources of acid in the Pittsburgh coal are several in number. These include pyritic concretions or sulphur balls, pyritic binders and top and bottom coals which have a higher sulphur content than the mined coal. In addition most of the strata up to the limestone or sandstone, as the case may be, are acid forming. It is suggested that the quantitative difference in acidity of the two parts of the Pittsburgh seam described are caused by the differences in roof structure rather than to the coal. All of the strata above the coal to the west of the Monongahela are highly acid forming. This is shown by their high content of oxidized sulphur in old areas. Table I gives the result of an analysis made on a series of strata in a mine which has produced much acid water but in a section which is dry at the present time. Ordinarily no appreciable sulphate sulphur is found in fresh samples. In this case where an old fall and the absence of rock dusting had allowed oxidation to proceed, the sulphate sulphur varied from 0.37 to 2.72 pct in the exposed strata. The lowest value of 0.37 pct is for the dense shale known as black rock which commonly forms the anchor for roof bolts in this seam. Individual spots could be noticed where the acid salts, principally Starkeyite (ferrous sulphate tetrahydrate) and Alunogenite (aluminum sulphate) were developed in masses of crystals, almost obscuring the parent rock.

Table I. Sulphate Content of Exposed Roof Strata

Sample	Distance Above Top of Seam, In.	Pct $SO_4^{=}$—S
Shale	5	0.58
Rider coal	21	0.87
Shale	35	1.17
Bony coal	38	2.72
Rider coal	48	0.88
Dense shale	60	0.37

In new workings situated so that the possibility of acid water backing up and coming down as roof drips is remote, acid roof drips were frequently noticed. However, they occurred only where a roof fall had exposed the immediate overlying strata. Water dripping through intact roof and roof coal was not acid. Wet or damp areas containing numerous sulphur balls were not found to be acid. The damp surface of a sulphur ball exposed for 3 weeks to air did not have an acidic reaction. A nearby recent roof break had drops of acid water.

The general results of these observations seem to be that of the many sulphuritic materials associated with the Pittsburgh seam, the most readily and rapidly oxidized are the high sulphur shales and rider coals. The exact nature of this material is not known but it may be finely disseminated pyrite. There is no doubt that sulphur balls, if left underground, in time oxidize to ferrous sulphate. In the Pittsburgh seam, which is the greatest acid producer, the vast extent of potentially acidic roof strata probably is the most important single factor in acid formation. In other seams the roof coals and shales are neither as great in quantity nor as uniformly acid forming. It would be informative to investigate acid forming and nonacid forming shales such as those in the Upper Freeport with a view to chemical composition and the behavior of sulphur oxidizing bacteria.

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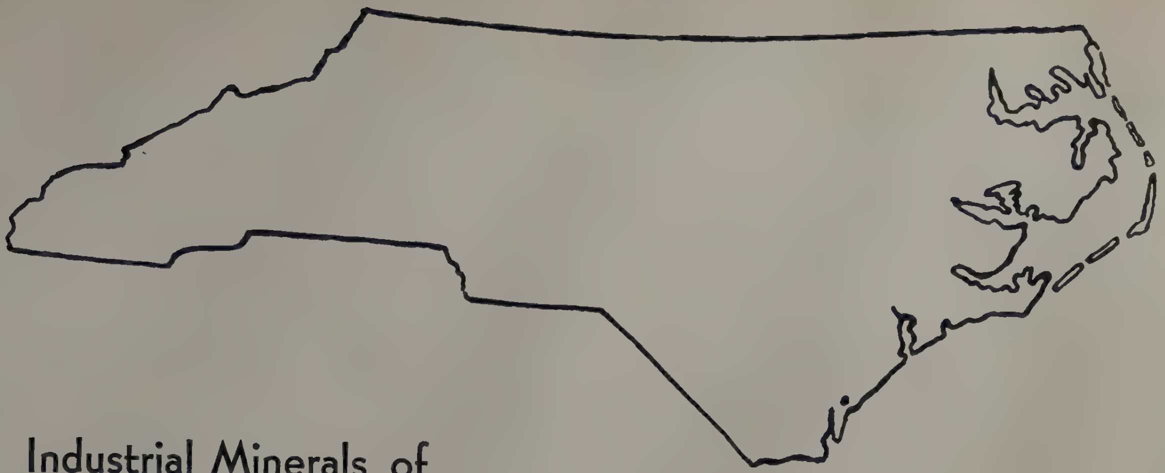

Industrial Minerals of North Carolina

by Jasper L. Stuckey

Geological investigation and research have contributed greatly in making industrial minerals the basis of an important industry in the state. North Carolina contains a wide variety of industrial minerals and rock. Mica, feldspar, kaolin, halloysite, talc, pyrophyllite, sillimanite, and spodumene are used to illustrate the progress being made.

In a previous paper[1] an attempt was made to indicate the importance of industrial minerals in North Carolina and the wide variety of such materials present. The present paper is concerned with geological investigation and research which have made industrial minerals the basis of an important industry in the state.

Geological mapping has gone through three stages in North Carolina during the present century. The first stage is represented by excellent reports of which *Corundum and the Peridotites of Western North Carolina*,[2] Volume I, now a classic, is representative. In this report, a geologic map of the western third of the state is shown on a scale of 10 miles to the inch. The next stage is represented by folios[3] of the U. S. Geological Survey published between 1902 and 1932. Most of these contained 30 min quadrangle maps on a scale of 2 miles to the inch. The third step is represented by reports prepared during the past 10 years which contain geological maps[4] on scales varying from 500 ft to 1 mile to the inch, while many mine maps have been prepared on scales varying from 20 to 150 ft to the inch.

As the scale on which geological mapping has been done has decreased, the amount of information secured has increased. Pratt and Lewis did excellent work on the peridotites of western North Carolina, but their maps lacked details. Keith and others added

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to the information on peridotites in various folios of the U. S. Geological Survey, but a scale of 2 miles to the inch on a map gives little real information. It was not until detailed mapping was done by Hunter[5] and Murdock[6] that the peridotites were known to contain large deposits of olivine and important amounts of vermiculite, asbestos and talc.

Ries,[7] Keith,[8] Watts,[8] and Bayley,[9] because of a lack of detailed geological maps, stressed the small size of the pegmatites, and Bayley estimated the recoverable kaolin of the state at about 625,000 tons of refined product. The more detailed work by Hunter,[10] Olson,[4] and Parker[11] indicates that at least 225 sq miles in the Spruce Pine district are underlain with a fine-grained pegmatite or alaskite type rock which contains 50,000,000 tons of crude kaolin, large amounts of flake mica and almost unlimited reserves of feldspar.

Research

Investigation and research in mineral dressing and utilization have also played important roles in the development of an industrial mineral industry in the state. The first important work of this type was begun in 1936 when the Tennessee Valley Authority established a ceramics laboratory at Norris, Tenn., to work out improved methods of refining North Carolina kaolin. As a result of experiments carried out jointly with the U. S. Bureau of Mines and the Harris Clay Co. of Spruce Pine, N. C., a modern kaolin refining plant was built in the Spruce Pine district in 1938.

In 1945 a cooperative program was inaugurated under which the North Carolina State College, the North Carolina Dept. of Conservation and Development, and the Tennessee Valley Authority began operation of a Minerals Research Laboratory in Asheville in a building constructed specifically for that purpose by the state of North Carolina. A number of programs of mineral beneficiation have been carried on. Among the more important programs are: 1—recovery of feldspar from alaskite or granite by flotation, 2—bleaching stained talc to improve the color, 3—concentrating flake or scrap mica, 4—drying and grinding mica, 5—concentrating sillimanite, and 6—flotation studies on spodumene.

The industrial rocks and minerals of North Carolina which have become of greatest interest and importance during recent years include: granites and gneisses, clays and shales, sand and gravel, limestones and calcareous rocks, peridotite minerals, pegmatite minerals, and minerals associated with crystalline rocks, such as talc, pyrophyllite, sillimanite and kyanite. Since it is impossible to discuss all of these minerals in detail, the pegmatite minerals, talc, pyrophyllite, sillimanite and spodumene will be used as examples.

Feldspar

Feldspar production in North Carolina began in the Spruce Pine district with one producer in 1911, and by 1917 the state had become the leading producer, which position it still holds.

In the early days of the feldspar industry, production came largely from mica mines and small feldspar mines and prospects. As a result, standard grades and quality control were lacking, and much of the feldspar shipped was unsatisfactory. This led, in the early 1920's, to improved conditions under which larger companies were formed and mills for grinding feldspar were established in or near the Spruce Pine district. With these changes came methods of quality control which led to the abandonment of the smaller dikes and prospects and a search for larger feldspar bodies. It was soon discovered that large dikes with definite schist walls and bodies of feldspar in large masses of pegmatitic or graphic granite contained ample feldspar for enlarging the industry. A number of mines, among them the Deer Park, Hoot Owl, Chestnut Flats, Crabtree Falls and the McKinney, were developed in the Spruce Pine district, from each of which 50,000 to 100,000 tons of feldspar were removed over a period of years. While the major feldspar production came from the Spruce Pine district, worthwhile production has also come from many other mountain counties.

A second major development in feldspar production took place about the beginning of World War II when improved geological mapping demonstrated that large reserves of feldspar are present in fine-grained pegmatite, commonly known as alaskite,[10] in the Spruce Pine district. The knowledge that such materials existed in large amounts and the increased demands for feldspar, especially for use in the glass industry, lead immediately to concentration studies. In 1945 the Carolina Mineral Co., a division of the Consolidated Feldspar Corp., began operating at Kona a modern plant to produce feldspar concentrates by flotation. In 1946 the Minerals Research Laboratory, Asheville, began pilot plant research on the flotation of feldspar. Based on the results of this work, the Feldspar Flotation Corp., Spruce Pine, put into operation in January 1949, a modern $300,000 plant for the production of feldspar from alaskite. The first year's operation was so successful that additional equipment to increase output has already been installed. The United Feldspar and Minerals Corp. also began operation of a feldspar flotation plant in 1949, and plans are already underway to improve and enlarge this plant.

To furnish a feldspar of higher potash content than can be obtained from the alaskite, both the Carolina Mineral Co. and the United Feldspar and Minerals Corp. are adding to their mill feed material from large dumps which have accumulated at some of the older block spar mines in which high grade potash feldspar is abundant. The Feldspar Flotation Corp. is increasing its output of potash material by blending feldspar from alaskite with higher potash content feldspar produced by the Feldspar Milling Co. nearby.

About 40 pct of the alaskite going to the crusher is quartz that is recovered by flotation free of impurities. Much of this was formerly discarded as waste, but most of it is now being sold as potter's flint, and for glass making or for concrete and plaster. Most of the feldspar produced by flotation is for the glass trade with smaller amounts going for pottery. The demands for both uses are increasing. With changes due to production from alaskite, North Carolina has recently become the leading producer of ground feldspar.

Mica

The production of mica began in North Carolina in 1867. At first only sheets large enough to cut into patterns were of interest, but the scrap obtained from trimming the sheets soon became important for producing ground mica. With the advent of kaolin mining it was learned that much flake mica from a few inches to ½ in. in diam could be saved and made good grinding stock. The advent of feldspar mining in 1911 added supplies of both sheet and scrap mica. With increased demands for both kaolin and mica it was determined about 1916 that important amounts of mica less than ½ in. in diam could be saved in the process of washing kaolin. During the early 1920's interest turned to mica schist, and a considerable amount of this material was used for its mica content.

In 1938 it was discovered that many square miles in the Spruce Pine district[10] are underlain with a fine-grained pegmatite or granite type rock, that contains important deposits of partly altered pegmatite rock in which important amounts (10 to 20 pct) of scrap mica are present. Recent geological studies indicate the presence of such materials associated with pegmatites in other districts than Spruce Pine. The mining of these deposits has increased production of ground mica in the state.

Formerly all mica flakes smaller than +8 mesh were discarded in the older type working plants. Investigation in the Minerals Research Laboratory, Asheville, during the past 2 years has shown that approximately 50 pct of all the mica in many deposits is —8 mesh in size. A process, involving the use of Humphreys spirals, has been worked out for saving this fine mica, thereby practically doubling scrap mica reserves of the state. Both wet and dry ground mica are produced, but the improved quality from the washers is making possible a dry ground product which is beginning to compete with the more costly wet-ground mica.

The drying of washed scrap has been a major problem in the industry for years. Driers were first heated with wood, and later by stoker-fired coal burners. More recently oil has become important, and now a strong effort is being made to bring natural gas into the area. Much of the mica is dried and bagged for shipment to the grinding plant. One important producer of washed mica had been shipping by rail. The drying plant of this producer burned, and shipment of wet mica in coal cars was tried. It was found that the vibrations of the cars en route dewatered the mica to such an extent that a smaller dryer at the grinding plant could be substituted for the larger and more costly plant which had burned. While North Carolina has been recognized for many years as an important producer of sheet mica, this industry is unstable and varies widely with economic conditions. Scrap mica, all of which is sold as ground mica, has shown a steady increase in production for many years and has more than trebled since 1940.

Halloysite

In the early days of kaolin mining in North Carolina between 1888 and 1900, some of the deposits in Jackson County furnished a clay that did not conform in all physical properties to true kaolin. Recoveries of as much as 40 pct of kaolin were reported from some deposits. Recent work by Hunter[12] has demonstrated that considerable amounts of that clay was halloysite and also that halloysite is not uncommon in the kaolin deposits of the state.

The presence of halloysite in important amounts in association with kaolin was first established in the early 1940's and halloysite mining actually began in 1943. It is now known that halloysite occurs in two main types of deposits: one of these is block halloysite which occurs as rich pockets in pegmatite dikes, and the other is halloysite mixed with quartz and clay as lenses in the large kaolin deposits in the alaskite or granite bodies. The first type is widely scattered throughout the pegmatite-bearing portions of the western part of the state. The second is restricted to kaolin deposits associated with the alaskite or granite of the Spruce Pine district.

Block halloysite is mined by opencut methods from the pegmatite bodies. The pure halloysite is hand-cobbed as blocks of varying size which are inspected and hand-trimmed to remove all iron stain. Following this the trimmed blocks are air dried and sold. This process is used only to a limited extent. In the larger kaolin deposits associated with alaskite or granite, the halloysite-bearing lenses are left until enough is secured to justify mining. These lenses are then mined, transported to the clay washing plant, and refined just as kaolin is refined. Halloysite mining and concentrating is costly as compared with kaolin, but the price is higher. Production of halloysite is increasing rapidly in the state.

The fact that detailed geological investigation and research on the preparation and use of pegmatite minerals have brought about rapid advances in their production in North Carolina can be best illustrated by comparing the 1940 production value of these minerals with their January 1950 production value. In 1940 the total value of all pegmatite minerals produced in the state was reported as $1,020,863. Reliable and conservative estimates indicate that the January 1950 production value of pegmatite minerals in the Spruce Pine district alone was at the rate of $6,000,000 annually.

Sillimanite

Sillimanite has been known as a rock-forming mineral in North Carolina since 1873, but it was not until 1943 that deposits of potential value were reported. In that year Charles E. Hunter,[15] TVA Geologist, discovered extensive outcrops of sillimanite-bearing rocks near the town of Valdese and in the South Mountains of Burke County. During the summer of 1945 the occurrences were examined and proved to be more extensive than was expected. A zone approximately 10 miles wide and 95 miles long and extending from near Elkin, Surry County, to Cliffside, Rutherford County, was outlined in which considerable sillimanite was found.

During the following 3 years interest increased in sillimanite, and new deposits were reported. Some concentration studies were carried out in the Minerals Research Laboratory, Asheville, but the results were unsatisfactory partly because of the highly weathered conditions of the samples.

By the summer of 1949 sufficient interest had developed to justify further study. A more detailed examination of all reported occurrence was made, and samples were collected for concentration studies. In addition to the zone extending from Elkin to Cliffside another zone was discovered which begins in Jackson County, crosses Macon County north and west of the town of Franklin and extends into Clay County.

Two types of sillimanite occurrences were observed. One of these consists of irregular masses of coarsely crystalline sillimanite varying in weight from 1 to 2 lb to as much as 250 or 300 lb. Such masses contain approximately 90 pct sillimanite. The other consists of irregular lenses of sillimanite gneiss and schist that contain from a trace to 20 pct sillimanite.

On the basis of work done to date, both the quantity and quality of sillimanite in the state appear promising.

Talc

Talc mining began in North Carolina in 1859 and has continued intermittently since. A few mines, including the Hewitt, the Hayes, and the Kinsey, were operated for several years with reasonable success, but because of a lack of geological information much useless prospecting was done. In 1945 detailed geological mapping of the area was begun on a scale of 1:24,000.[13] It is now known for the first time that adequate reserves of talc for long term operations are available. A process for reclaiming iron-stained talc has been worked out—further increasing the potential reserves. The talc mining and preparation industry of the state has been recently described by Van Horn.[13]

Pyrophyllite

Pyrophyllite deposits have been known to occur in North Carolina since 1856, but their true value was not realized until the publication in 1928[14] of a report containing detailed geological information on the deposits and a modern geological map of the area. Mines and mills are now being operated at Robbins and Glendon, Moore County and Staley, Randolph County. Pyrophyllite is being mined near Snow Camp, Alamance County and manufactured into refractories at Pamona, near Greensboro. These refractories are being shipped as far as Chicago. Pyrophyllite mining has become of major importance in the mineral industry of North Carolina in recent years and constant search is being made for new deposits.

Kaolin

Modern kaolin mining in North Carolina was begun near Webster, Jackson County, in 1888 and for some 20 years was carried on chiefly in Jackson, Swain, and Macon counties. All the kaolin bodies in that area are in relatively small pegmatites, and mining was carried on from open cuts for the first 15 to 20 ft followed by circular type cribbed shaft for greater depths. Mining was carried out entirely by hand labor. When kaolin mining shifted to the Spruce Pine district, between 1900 and 1910, where there are large deposits derived from alaskite, open-cut mining became standard practice. Owing to the topographic position of many of the deposits, hydraulic mining replaced hand labor in getting the clay into the flume lines. As the pits became deeper, bucket elevators came into general use. In the early 1920's attempts were made to substitute mechanical classifiers for the old machinery, and gradually processes in use today became established.

Today, kaolin mining and concentrating is completely mechanized. At the newest washing plant the clay is ground in Hardinge ball mill and concentrated in a cone classifier. This has increased the recovery by disintegrating the coarser particles, many of which were lost in the older processes, and has made available finished kaolin of a much higher quality than was produced formerly.

Spodumene

It has been known for many years that small amounts of spodumene occur in pegmatite dikes in the Kings Mountain district. Little interest was shown in these occurrences until the publication in 1942 by the U. S. Geological Survey[16] of a report containing detailed geological maps and careful estimates indicating the presence of 650,000 tons of recoverable spodumene in rock averaging 15 pct spodumene. During World War II the Solvay Co. built a flotation plant at Kings Mountain and produced spodumene with some success for the war effort. At the end of the war the plant was closed and has remained idle since. Early in 1949 the Foote Mineral Co. leased mineral lands adjoining the Solvay holdings and later leased the Solvay property. Detailed flotation studies on concentrating spodumene and on iron removal from spodumene concentrates are being carried out in the Minerals Research Laboratory in Asheville. A series of chemical analyses indicate almost a theoretical and an unusually uniform lithia content in the spodumene throughout the district. Plans are being developed to produce spodumene for the ceramic industry as well as for lithium compounds.

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Operational Studies in the Pennsylvania Slate Industry

by W. F. Mullen and C. W. Stickler

WITH few exceptions, unit operations in the Pennsylvania slate industry in 1950 did not differ appreciably from production methods described by Behre[1] and Bowles[2-4] several decades ago. Many traditionally picturesque but relatively inefficient hand operations continued to contribute to high operating costs in an industry in which the margin of profit, for various reasons, was admittedly low.

As part of a general program in which the Pennsylvania State College has been assisting the slate industry to solve some of its problems, time and method studies were used in an effort to determine the bottlenecks of production flow. The results are as applicable to other stone industries as they are to slate.

A study of the basic elements in the production of roofing slate, structural slate, and slate blackboards among selected slate producers revealed comparatively few refinements. Any marked difference in operating method was characterized largely by the available equipment, a noticeable difference in the working properties of the rock quarried by a particular company, and by either the habit thinking or ingenuity of the individual operators.

With the hope that significant economies in manufacture might result, all the unit operations were broken down into elements for study and analysis. The greater part of the study encompassed those operations from the quarry rim to the finished product; however, to obtain a reasonable synthesis of time allotments per operation, several analyses of quarrying from the finish of the wire saw cut are included for reference. The complete study of quarrying made by Bowles several years ago resulted in the introduction of the wire saw into the operation; there appears to be a need for as fundamental a change in the processing of finished slate as the wire saw was in the quarrying process.

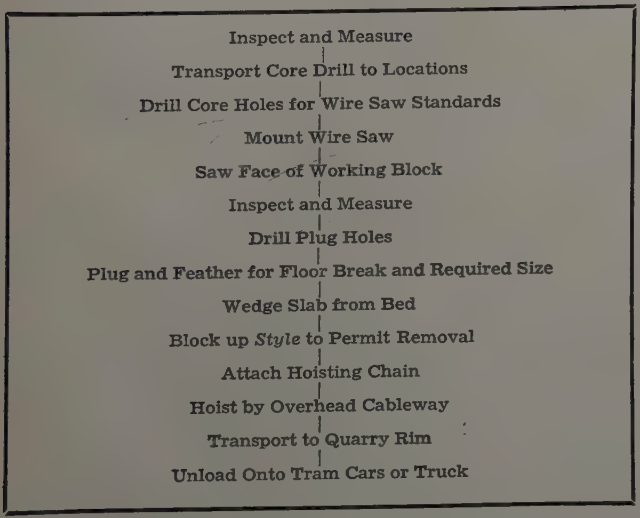
Inspect and Measure
Transport Core Drill to Locations
Drill Core Holes for Wire Saw Standards
Mount Wire Saw
Saw Face of Working Block
Inspect and Measure
Drill Plug Holes
Plug and Feather for Floor Break and Required Size
Wedge Slab from Bed
Block up *Style* to Permit Removal
Attach Hoisting Chain
Hoist by Overhead Cableway
Transport to Quarry Rim
Unload Onto Tram Cars or Truck

Fig. 1—Flowsheet of quarrying operation.

Fig. 1 illustrates the flowsheet of quarried rock from the parent bed to the quarry rim with the basic elements which contribute to the difficulty of removing slate from the ground.

In Pennsylvania roofing slate is produced by one of three procedures, many operations of which are quite similar. In the traditional or classical method the quarried rock is reduced by sculping and auxiliary manual treatment to workable size and then is split and dressed (trimmed to size) in the conventional shanty. Archaic as this method might appear, there are several elements that result in substantial savings over more mechanized operations.

In a modified version of this method, the block is reduced by sawing and gouging preparatory to splitting and dressing in indoor stations. This method is particularly adaptable to rock which fractures unevenly by conventional sculping methods.

In the third and more modern method, reduced block received from the block maker is diamond-sawed to length and finished on a production line. Fig. 2 illustrates the flow pattern in each case.

In the modified and more modern plans, dressing is done on mechanically-operated trimming machines, which appreciably reduce the fatigue factor of the operator. However, it should be noted that it is advantageous for the operator to be able to control the speed of the knife to prevent breakage of certain classes of stock. In the classical plan the splitter and dresser normally act as blockmakers also and carry their *eighter* slabs (equivalent in thickness to eight shingles) into the shanties themselves, all of which adds to fatigue and reduced efficiency. The extremely low investment and operating expense of the classical method has undoubtedly been of paramount importance in its continuation.

Production of Mill Stock

The production of mill stock is accomplished by less picturesque methods than enter into the production of roofing slate but it is, in the main, relatively more efficient because of the increased use of machinery for handling and for finishing.

Mill stock can be classified into two categories as far as difference in production method is concerned: structural slate and blackboards. The significant difference in operation is caused largely by the nature of the rock used for each product with the best and easiest cleaved rock being reserved for blackboards to permit the splitting of large slabs to as little as ½ in. in thickness. Structural stock, on the other hand, is split into panels no thinner than 1½ in. and

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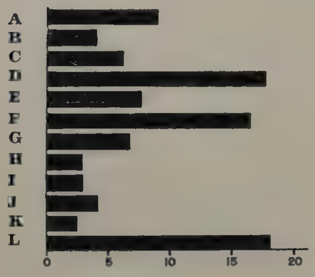
Table I. Quarrying

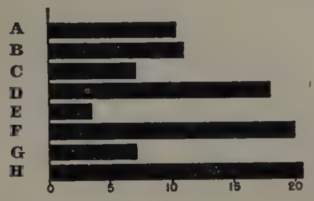
	Element	Average Time, Min
A	Inspect and measure	2.35
B	Drill plug holes	1.05
C	Plug and feather	1.57
D	Wedge slab from bed	4.65
E	Style	2.00
F	Signal and wait for tackle	4.35
G	Attach chain to tackle	1.73
H	Position block	0.72
I	Hoist, 75 ft	0.75
J	Transport to rim, 100 ft	1.02
K	Unload	0.61
L	Incidentals, look for and put away tools, wait for help, remove waste, etc.	4.75
		25.55



Relation of elements to unit operation, pct.

Table II. Block-Making

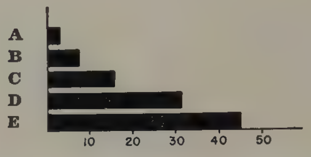
	Element	Average Time, Min
A	Inspect and measure	3.71
B	Drill plug holes, three	4.03
C	Line holes	2.49
D	Plug and feather	6.58
E	Bar and turn over	1.24
F	Split into eighters	7.33
G	Carry to splitting station	2.58
H	Incidentals, swab slab, look for tools, trim clean-up, etc.	7.66
		35.62



Relation of elements to unit operation, pct.

Table III. Splitting 3/16 In. Shingle

	Element	Average Time, Min
A	Pick up eighter	0.05
B	Split to ½'s	0.16
C	Split ½'s to ¼'s	0.33
D	Split ¼'s to ⅛'s	0.68
E	Measure shingle, pick to desired length, and remove to storage for dresser	0.97
		2.19 per eighter

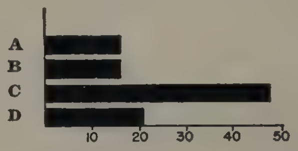


Relation of elements to unit operation, pct.

Table IV. Dressing

	Element	Average Time, Min
A	Pick up slate	0.03
B	Make[a] slate	0.03
C	Measure and trim	0.09
D	Stack on rack	0.04
		0.19 per shingle

[a] Determining the proper size finished shingle that can be obtained and making the initial trim on the dressing machine.



Relation of elements to unit operation, pct.

normally is characterized by a rough split, which requires a planing operation in addition to the surface treatment given to blackboard stock. Considerable attention was devoted to the planing operation during our study inasmuch as it appeared that fundamental improvements in the milling sequence would hinge largely on the planing operation as a nucleus.

Fig. 3 illustrates the flow pattern for both structural slate and blackboard production. Under the category of *special finishing* of structural slate are included such manual operations as chamfering of edges, sawing or planing of grooves, and the drilling of holes.

Supplementary storage and consequent rehandling operations not shown on the flow pattern occur frequently, particularly when normal flow is interrupted and slabs accumulate at a given station. The handling problem is particularly acute inasmuch as the only methods currently employed are those involving trundling, carrying on the operator's back, wheeling on a two-wheel cart, or, in a few exceptional cases, the use of a manually operated pneumatic hoist.

Time Study

The time study was employed as a means of analyzing the present production pattern for purposes of determining the contributory factors in the present relatively inefficient methods for finishing slate. There was no intent to establish standards or to become involved in the various ramifications of cost control. The purpose of the data is to direct attention to those elements or operations that appear to be most in need of correction; future and more detailed studies can be employed to complete the picture.

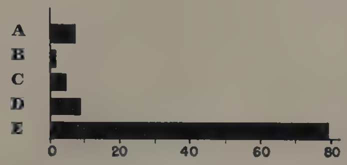
Tables I through VIII show representative time for the individual elements of each unit operation in addition to showing the percentile relationship of each element to the total operation and each unit operation to the total amount of work required to produce the finished product.

More detailed analysis of each element and operation falls outside the scope of this discussion but is contained in a forthcoming bulletin of The Pennsylvania State College Mineral Industries Experiment Station.

Tables IX and X tabulate the comparative amount of time required per unit operation. Table IX covers the production of 160 roofing shingles from an orig-

Table V. Sawing

	Element	Average Time, Min
A	Load	6.10
B	Unload	1.15
C	Brace	4.32
D	Wedge sawed sections	7.95
E	Saw four sides	75.40
		94.92

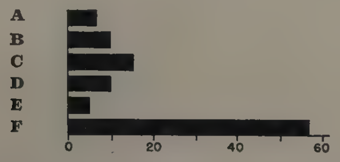


Relation of elements to unit operation, pct.

Table VI. Planing

	Element	Average Time, Min
A	Load	0.94
B	Brace	1.50
C	Incidentals[a]	2.20
D	Turn over	1.50
E	Unload	0.60
F	Plane both faces	8.86
		15.60 per slab

[a] Align knife, adjust machine, clean up, etc.



Relation of elements to unit operation, pct.

inal block 172 in. long, 32 in. wide, and 9 in. thick with transport time from quarry rim to station and punching time for roofing shingles included to complete the study. Table X covers the production of six structural slabs each 33½x53x¾ in. thick from a block sawed on four sides to these linear dimensions. The original thickness of this block before splitting was 9⅞ in. Transport time from quarry rim to mill and splitting time for the slabs are included to complete the study.

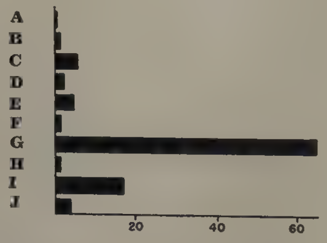
Analysis of Results

A study of the various time summaries obtained reveals one troublesome factor: each block removed from the quarry possesses inherent characteristics that affect the handling methods and operational time. In planning flow patterns it is essential that methods be devised to nullify these variations through improved mechanical equipment or to allow flexible routines in the milling operations.

Handling time poses problems in the way of distance traveled, weight and size of block, condition of roadbed, and type of hauling unit. Weather conditions and safety regulations also play dominant roles in anomalies from normal conditions. Such variations appear to be sufficiently frequent to preclude any standardization from data obtained at this stage of the study. As yet no reproducible relationship has been formulated to apply to the many plants studied in the industry but the results of our study have indicated definitely the possibilities of improved methods, equipment, and routines that may be applicable to the industry as a whole.

Table VII. Sand Rubbing

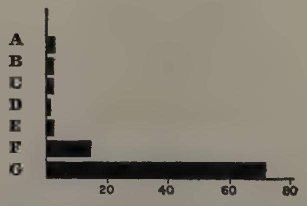
	Element	Average Time, Min
A	Position slab horizontally	0.25
B	Place weights on slab	0.49
C	Change weights	1.10
D	Reposition board	0.42
E	Gage and remove weights	0.87
F	Turn over slab and replace weights	0.34
G	Rub faces	15.86
H	Position slab vertically	0.32
I	Rub edges	4.11
J	Unload slab from bed	0.77
		24.53 per slab



Relation of elements to unit operation, pct.

Table VIII. Polishing

	Element	Average Time, Min
A	Transport slab to polisher	0.34
B	Position slab on table	0.30
C	Touchup	0.25
D	Unload from machine	0.19
E	Remove from storage	0.29
F	Wash and inspect	1.37
G	Polish	7.49
		10.23 per slab



Relation of elements to unit operation, pct.

Table IX. Production of Roofing Slate

Unit Operation	Time, Min
Quarrying	25.55 (1 block)
Transport	9.70 (1 block)
Block-making	35.62 (20 eighters)
Splitting	43.80 (160 shingles)
Dressing	36.40 (160 shingles)
Punching	15.98 (160 shingles)
Total	167.05

Suggested Improvements

Based on the observations gathered in this study, certain conclusions were reached with respect to methods improvement, equipment design and better utilization of raw material.

It is apparent that major changes are necessary in present operating methods not the least of which would be the finishing of multiple pieces in one operation. These unit operations under consideration for this transformation are: 1—splitting, 2—dressing, 3—sawing, and 4—planing, rubbing, polishing. Composite machines to do the work of several opera-

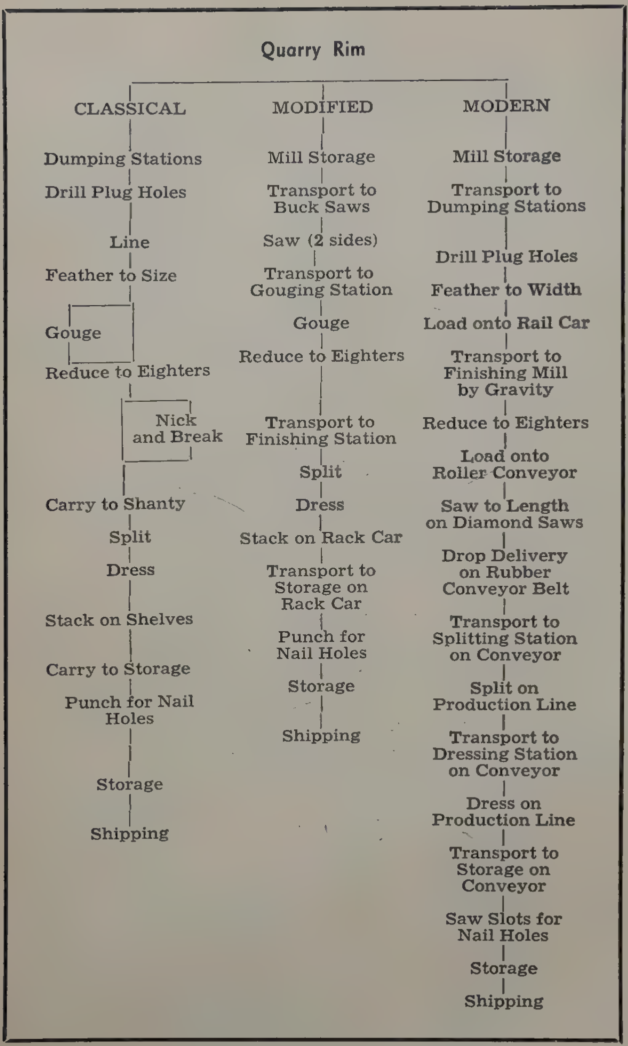


Fig. 2—Flowsheet of roofing slate production.

tions would materially decrease the amount of handling time between operations and reduce the amount of waste resulting from separate operations. Under study at the college are new designs in planing, rubbing, and polishing operations that will simplify material flow, increase the yield from quarried slate, and increase production.

There are promising indications that saw speed with the present design saw can be increased at least fourfold by the use of a new type inserted tooth now under test. Sawing with the present saw occupies such a prominent relationship to the overall time picture mainly because only one side of a block can be sawed at one time. Bowles[3] has referred to the use of gang saws in the Vermont-New York region for slate too hard to cut by conventional saws. Gang-sawing may be the answer to multiple side sawing of blocks although it may not necessarily take the form of the conventional bladed gang saw.

Table X. Production of Mill Stock

Unit Operation	Time, Min
Quarrying	25.55 (1 block)
Transport	6.13 (1 block)
Saw (4 sides)	94.92 (1 block)
Split	19.06 (6 slabs)
Plane	93.60 (6 slabs)
Rub	146.58 (6 slabs)
Polish	61.38 (6 slabs)
Total	447.22

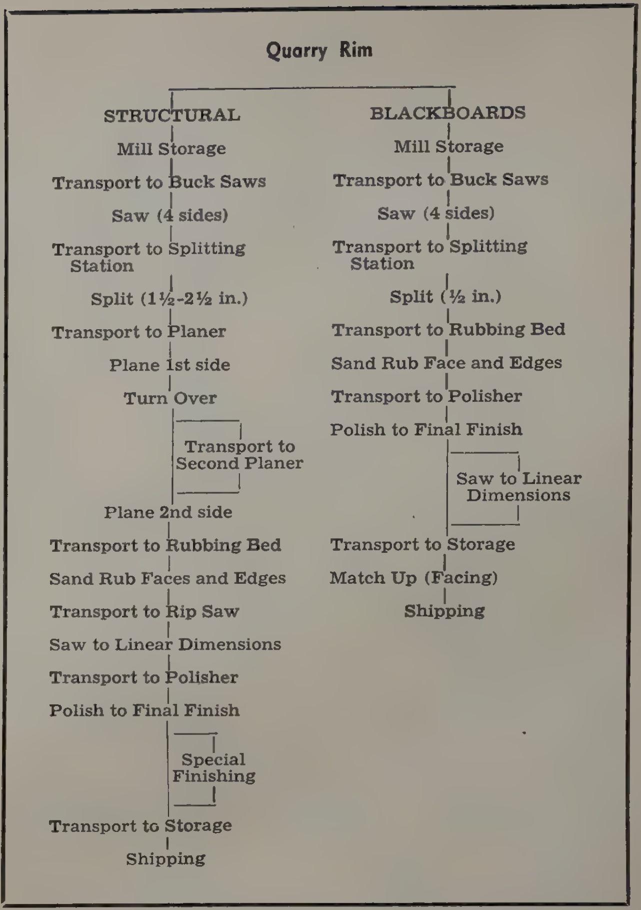


Fig. 3—Flowsheet of structural and blackboard production.

Under consideration at the present time are two possibilities: a multiunit wire saw for sawing two sides of the block simultaneously either with single blocks or blocks arranged in tandem (the French[5] have used wire saws for their milling operations rather successfully) and a multiunit diamond saw with speed reducer to control the load on the motor with the nature of the rock and the speed of sawing. Another possibility that could be developed to minimize the need for mill sawing would be the design and development of a quarry saw on the chain-saw or cutter bar pattern, which could saw blocks to mill size directly from the bed. In addition to eliminating the need for mill sawing, such a saw would eliminate most of the waste resulting from the ragged fractures, which are sawed off and discarded in the mill.

Also under study is the use of overhead monorail conveyor systems in the mill as a corrective for handling delays.

Further industrial studies appear to be necessary to expand present information on slate production. Sufficient attention to this overlooked industry should soon result in a production pattern that will permit lower operating costs and greater prosperity for the industry.

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Mine-Drainage Studies in the Iron Ranges of Northern Michigan

by Wilbur T. Stuart

THE increased demand for iron ore has necessitated a re-examination of ore-bearing lands on which the presence of water previously has indicated hazardous and expensive operating conditions. In view of the importance of iron ore production to the national economy and defense, the Ground Water Branch of the U. S. Geological Survey, in cooperation with the state of Michigan, began a study of mine drainage in the Iron River district in 1945, and later extended the work to the Marquette district. The purpose of these studies was to define the principles involved in the movement of surface and ground water toward the mined areas, with the hope that the information obtained in the research would lead to the development of methods of water control and to a reduction in the total mining costs.

Not only are the direct costs of drainage increasing, but also the indirect operational costs of working a wet mine are becoming a larger proportion of the total mining costs. For some wet mines the direct costs of pumping and drainage may range from 35 to 40¢ per ton of ore produced, but the indirect costs due to handling wet ore and controlling the water may be five to ten times this amount.

The methods of study were formulated as the work progressed, and inasmuch as they were the first large-scale studies of their kind, they should serve as a guide for the solution of similar problems in other mining areas. The Iron River district was chosen for a pilot study because in this district the longest records of mine pumpage and water-level observations were available, including, as they do, the records for the Homer mine of the M. A. Hanna Co., where pumping from surface wells began in 1930.

The results of the first investigation by the Michigan Department of Conservation have already appeared.* The first section of the report on a similar study of the Marquette district, which was started in 1948, will be published this year.

Methods of Study

In an effort to reduce the flow of water into the mine workings in the Iron River district, about 4500 gpm was pumped from the bedrock being mined and about 9000 gpm from the glacial overburden. In the Marquette district in the vicinity of Ishpeming and Negaunee, about 5000 gpm was pumped from the bedrock and about 4000 gpm was pumped from the glacial overburden. Where the water was pumped only from the bedrock, the rate of pumping ranged from a few hundred gallons per minute in the dry mines to many hundred gallons per minute in the wet mines. In each district, pumping from the overburden was localized on a few properties where costly pumping installations had been made and the expenditures for power had been large.

In each district a comprehensive ground-water investigation was made of the whole area, involving the collection and interpretation of all the available data bearing on the source and quantity of water to be controlled. Although it is not the purpose of this paper to discuss the methods of making a ground-water investigation, it should be pointed out that a drainage study follows a pattern of engineering analysis that determines the occurrence, source, movement, disposal, and quantities of water involved. The investigation in the iron-mining districts of Michigan began with the construction of a map of the buried bedrock topography. Because the ground-water reservoirs occupy the low points in the bedrock basins, this map gives information concerning their areal extent, depth, shape, and degree of interconnection. The depth to water in the drillholes and wells indicates the altitude to which the ground-water reservoirs are filled, and the logs of the material penetrated in the drillholes and wells indicate the general character of the materials filling the reservoir. The slope of the ground-water surface indicates the direction of flow through the reservoir, the water movement being from the points of higher altitude to points of lower altitude. By analysis of the rise and fall of the ground-water levels in response to additions of water through recharge and to changes in the rate of discharge through pumped wells, estimates of the total quantity of water in storage and of the rate of flow through the reservoir

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* W. T. Stuart, C. V. Theis, G. M. Stanley: Ground-water Problems of the Iron River District, Mich. Technical Report No. 2, Geol. Survey Div., Mich. Dept. of Conservation.

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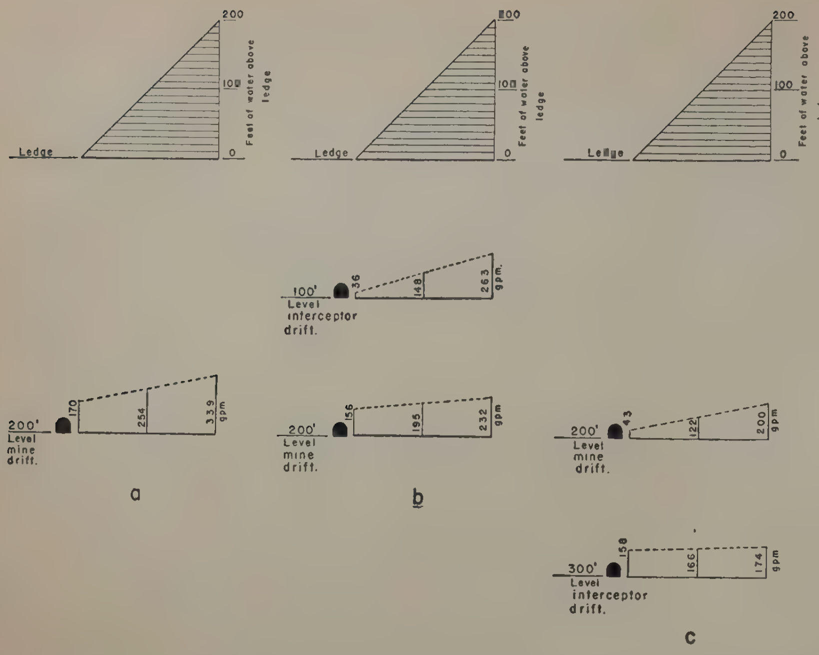


Fig. 1—Relative discharge, in gallons per minute, from mine and interceptor drifts 1000 ft long, 16 ft wide, at footwall, at selected levels in a vertical (iron formation) aquifer 157 ft wide.

can be made. When the principles of movement of the ground water to the mined areas are understood the most feasible means of control can be selected.

Recharge

The ground-water reservoirs in northern Michigan are replenished or recharged by precipitation and in certain areas by the induced percolation from the surface streams and ditches. Through studies of the records of the water-level fluctuations in the wells and drillholes it has been determined that approximately 20 pct of the total precipitation recharges the ground-water reservoir, which represents, in a normal year, an equivalent of about 200 gpm per sq mile. In the Mineral Hills area of the Iron River district the recharge induced from the surface streams and ditches within the cones of pumping depression may exceed an average of 1000 gpm. In the Marquette district the recharge induced from the Carp River near the Morris mine of the Inland Steel Co. probably averages about 500 gpm.

Ground-Water Movement

In the zone of circulation above the bedrock the ground-water movement is from the points of high altitude, where replenishment is taking place, to the points of low altitude where the ground water is discharged at the land surface. The gradients are normally low, and the velocity of water is also low, even though the water-bearing materials may be able to transmit a large amount of water. When the altitude of the discharge point is artificially lowered, such as in the cone of depression around a pumping well or other drainage structure, the gradient is increased greatly, and the quantity of water flowing to the structure is increased greatly. When an active mine lowers the water level through mining operations, the gradient is increased, and a cone of water-table depression or a ground-water sink is formed in the saturated glacial overburden above the point at which the water is descending into the mine workings. The drainage downward through this ground-water sink produces the same hydraulic result in the glacial overburden as a vertical well discharging an equal quantity of water, and the same mathematical formulas apply in both cases to the rate and extent of the cone of depression. These formulas indicate that the depth of the cone varies directly as the quantity of ground water withdrawn; and the increase in diameter of the cone varies directly as the time since the withdrawal of water began, although the outward growth is faster when the process takes place in minerals of greater permeability.

The investigations in the Michigan mining districts show that pumping the ground-water reservoirs in the glacial overburden has little effect on the flow of water into the bedrock, so long as there is a great thickness of water over the bedrock openings. Pumping from the overburden is most effective where the areas of permeable bedrock can be dried up by drawing the water table down so as to cause the ground-water shore line in the overburden to retreat down the bedrock slope, either uncovering the bedrock or reducing the head of water above it to a negligible amount. It was further determined during the investigation that, where dewatering is attempted by means of wells, they should be located as near as possible to the point at which the water enters the bedrock.

There are two manners in which water from the ground-water reservoirs in the glacial overburden can move into the bedrock: 1—by slow percolation through the capillary openings in the bedrock, 2—and by rapid movement through supercapillary openings. In the first case the rate of flow of water through the bedrock itself governs the quantity of water in the mine, and generally the flow is of the order of a few hundred gallons per minute. In the second case, where there is rapid movement in supercapillary openings such as vug systems or cracks formed by the collapse of the roofs of mine workings, the rate of flow into the mine may be governed by either the rate of flow in the bedrock or, in case the openings are of larger size and of greater number, the rate at which the overburden can transmit water to the area. In either case, because of the rapid movement through large openings, the mine would be wet and the quantity of water pumped from it would be many hundred gallons per minute.

In the Michigan iron mining districts the term iron formation is applied to that particular structural member or bed which carries the minable percentages of iron ore. In the Iron River district the unoxidized part of the formation consists of interbanded chert and siderite, and the oxidized part consists of chert interbanded with soft red and yellow oxides, mostly hematite. In the Marquette district the unoxidized part of the formation consists of iron-bearing slates, schists and cherts, and siderites, and the oxidized part consists mostly of hematite with small widely disseminated particles of magnetite. In both districts the footwall and hanging-wall formations are more resistant members and are not ordinarily considered to be water yielding. Both the footwall and hanging-wall formations in the Iron River district are slate members, whereas in the Marquette district only the footwall is a slate. In the Marquette district the hanging-wall may be either a quartzite or a conglomerate.

A study of the movement of ground water in the orebodies indicates that the water enters the iron formation through its surface of contact with the overlying glacial blanket. This is shown by several lines of evidence. First, most of the water enters the mines through the workings in the iron formation. Water entering the drifts of the footwall and hanging-wall strata is of minor amount. Second, the water in the operating mines is of the same chemical character as the waters in the overburden, whereas the waters in the hanging-wall and the footwall strata are more mineralized and of a different chemical character. Third, those mines where the iron formation has its surface contact with the overlying glacial blanket either above the general ground-water level or in areas of small ground-water drainage are relatively dry mines.

The theoretical problems involved in the movement of ground water in the bedrock are more complex and of a different nature from those in the overburden. They differ in principle in that the movement in the bedrock here considered is in a bed or an aquifer that stands vertically, or steeply inclined, whereas the movement in the glacial overburden is essentially horizontal. The water entering the iron-bearing formation through its surface contact with the overlying glacial blanket moves downward essentially under gravitational force. The movement is complex, because the structural control is complex. In many places the effective boundaries of the ground-water reservoir above the iron formation and the structure of the iron formation from the bedrock surface down to the mine levels are unknown. Even where these factors are known they are often too complex to be amenable to mathematical treatment without considerable simplification. Such simplification of complex data requires analysis to obtain feasible approximations of actual conditions. However, one does not throw away a useful tool simply because one wishes it were better.

Some indication of the magnitude and complexity of the problems involved may be afforded by considering the movement of water in a new mine drift or an opening in an iron formation in a new locality. Presumably before the opening is made the water moves slowly. It enters the iron formation through the bedrock surface and moves to a lower point under a slight hydraulic gradient equal to the difference in the altitude between the two points divided by the total length of the formation between them. When the opening is made, a rapid movement of water begins. If the glacial drift blanket is saturated so that the iron formation is furnished all the water it can transmit and the dip of the formation is constant between the mine workings and the bedrock surface, the water will move under a hydraulic gradient equal to the sine of the angle of dip. Mathematical equations have been set up to determine the pressure head and direction of flow of water in an orebody under some of these conditions. These equations indicate that the water descending through the formation is under considerable pressure near the level of the lowest mine development and that the pressure head diminishes greatly at some distance above. The equations furnish a means of estimating the water to be expected in workings at several levels, especially in water-drainage drifts intended primarily to intercept descending water in the formation at a higher level than the main mining operation. In general, for nearly vertical formations it appears desirable to construct exploratory or preliminary openings and to collect most of the water at lower levels, for the cost of lifting water from a lower level is offset by the economy of mining a drier orebody above.

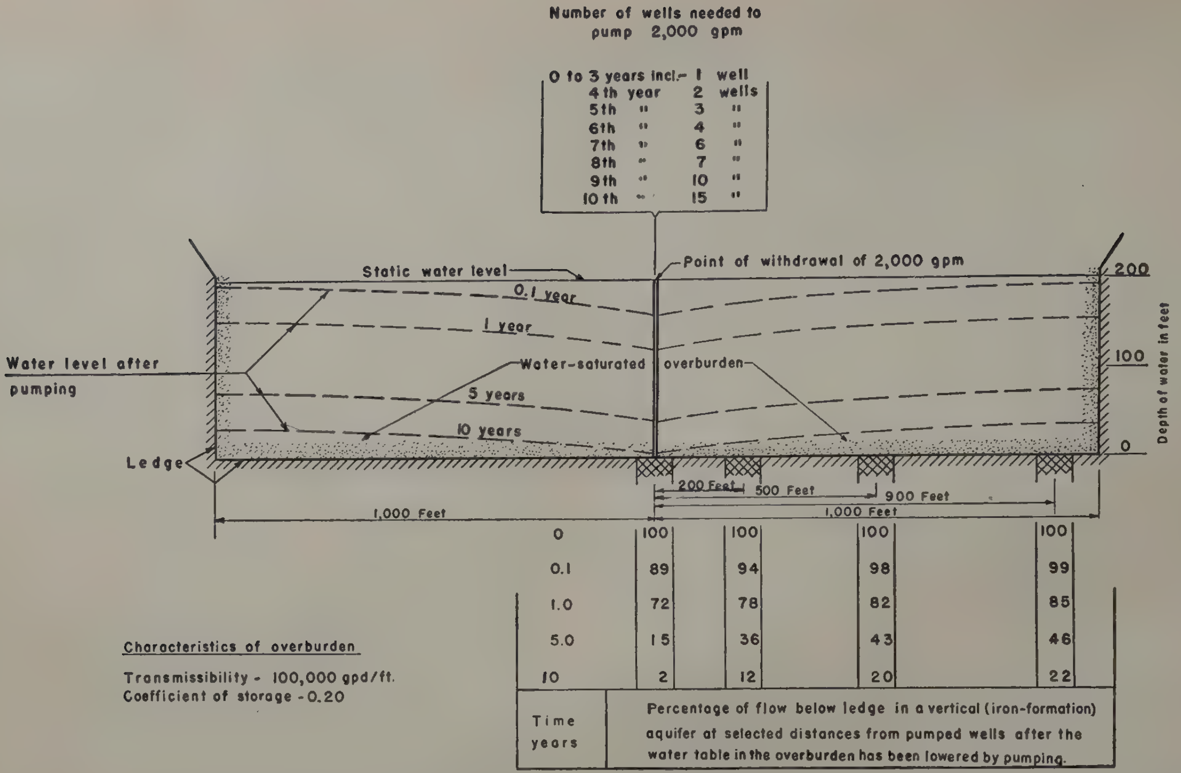
In case of an inclined or dipping formation, other equations are needed, as the pressure head is either increased or decreased by the gravity component of the downward movement of water which directs the flow toward the drift opening. In a monoclinal or rolling formation it is possible for a properly located drift at an upper level to divert all the water from the formation, especially if it is located along the saturated footwall of the structure.

Water Control Methods

Most simple cases of ground-water flow can be approximated mathematically, but others require long tedious computations. Since most mine operators are interested principally in how to control the water and in the amounts that must be handled, the quantities of water that could be expected from the two common methods of drainage or control are given in Figs. 1 and 2. The equations for the calculations are discussed in Mich. Geol. Survey Technical Report No. 2; pages 37 to 41 for Fig. 1, and pages 46 to 50 for Fig. 2.

Fig. 1 shows the flow of water that might be expected from mine drifts in the bedrock with various thicknesses of water in the overburden. In case *a* with a mine drift 200 ft below the bedrock, the flow of water doubles when the head of water in the overburden is increased from a negligible thickness to 200 ft. In case *b* with a main operational drift 200 ft below the bedrock surface and a water interceptor drift 100 ft below the bedrock surface (water is withdrawn from both drifts) the interceptor has little effect on the lower drift when there is a negligible thickness of water in the overburden; but with 200 ft of water in the overburden the total water at the operations level is decreased about 30 pct. The general efficacy of intercepting galleries varies greatly with the height of water above the bedrock. In case *c* in which the interceptor is placed at the 300-ft level and a negligible thickness of water above the bedrock is assumed, the water in the main operations level at 200 ft is reduced about 75 pct although the total cost of lifting the water is increased. Thus it is apparent that exploratory openings at lower levels would produce drier working conditions for mining operations above.

Fig. 2 shows a cross-section of an idealized buried valley being dewatered, and an iron-formation suboutcrop below the overburden. This cross-section, with its included tables, graphically demonstrates


Static water level
Point of withdrawal of 2,000 gpm
0.1 year
1 year
5 years
10 years
Water level after pumping
Water-saturated overburden
Ledge
200 Feet
500 Feet
900 Feet
1,000 Feet
1,000 Feet
200
100
0
Depth of water in feet
Characteristics of overburden
Transmissibility - 100,000 gpd/ft.
Coefficient of storage - 0.20

Number of wells needed to pump 2,000 gpm

0 to 3 years incl.	1 well
4th year	2 wells
5th "	3 "
6th "	4 "
7th "	6 "
8th "	7 "
9th "	10 "
10th "	15 "

Time years	Percentage of flow below ledge in a vertical (iron-formation) aquifer at selected distances from pumped wells after the water table in the overburden has been lowered by pumping.			
0	100	100	100	100
0.1	89	94	98	99
1.0	72	78	82	85
5.0	15	36	43	46
10	2	12	20	22

Fig. 2—Cross-section through an idealized buried valley showing relation of water table in the overburden to flow in formations below ledge.

typical events that occur whenever the overburden is dewatered by means of wells.

To indicate the percentage of reduction in flow through a vertical iron formation below a bedrock valley, at the well, and at distances of 200, 500, and 900 ft therefrom, a table of percentage of the original flow is shown for each time period. The greatest reduction in flow takes place where the water is lowest, and this point is over the point of entry of the water—the suboutcrop of the iron formation.

In the computations for Fig. 2 there was assumed to be 200 ft of saturated overburden above the bedrock and a constant withdrawal of 2000 gpm from the valley. No trouble would be experienced in maintaining this constant rate of withdrawal during the first 3 years of pumping, but after that time the decreasing thickness of saturated materials would require installation of additional wells to maintain the 2000 gpm rate. In most pumping installations the well systems dewater the area until the reduced yield results in no further lowering of the water table. Additional wells added to the system result in additional lowering and additional reduction in yield per well unit, and again the system reaches a new equilibrium. At the top of Fig. 2 a table indicates the approximate number of wells needed to maintain a constant yield of 2000 gpm which will lower the water table according to the time periods shown. For simplicity in this figure, it was assumed that all of the newly added wells would be located at the point of withdrawal; although the actual construction of the new wells either closely spaced in a line or a small circle would slightly modify the profiles.

Pumping directly from the glacial overburden requires the installation of expensive surface pumping equipment in an area beneath which mining operations are taking place, and where there is constant danger of damage to or loss of the installation by subsidence. A possible solution would be to encircle the critical area, in which the ground water enters the bedrock, with a ring of closely spaced small-diameter, low-cost wells. Pumping from these wells would lower the ground-water level in the reservoir almost to bedrock within the circle and thus reduce the water available for downward percolation. Because the wells must be closely spaced to reduce the saturated thickness of the water-bearing materials between them, the yield of any individual well would be small, thereby eliminating the need for expensive large-diameter wells. This low cost feature is important, because of the large number of wells needed to encircle the area and the likelihood that some may be lost owing to subsidence.

Other means of controlling water flows such as grouting, chemical seals, asphalt barriers, and freezing have not been tried on a large scale in Michigan, to the knowledge of the writer. However, in shaft sinking and in foundation work all have been used more or less successfully. Probably the high cost of the product, the large amount of drilling needed, and the unwillingness of any contractor to guarantee results have been the biggest factors in restricting the use of these methods, although it is believed that in the future these methods of control will be used with success.

After 5 years of study it is apparent that no one method or even combination of methods can be applied economically to eliminate all the ground water in mine workings. Each problem requires separate study and solution. Some properties may be effectively dewatered by pumping from the overburden. In others where surface pumping is not economical, properly located water-interception drifts may control the water so that a drier ore may be mined with fewer hazards and at less cost.

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